LEAD-SMELTING.

THE CONSTRUCTION, EQUIPMENT, AND OPERATION OF LEAD BLAST-FURNACES,

AND

OBSERVATIONS ON THE INFLUENCE OF METALLIC ELEMENTS ON SLAGS AND THE SCIENTIFIC HANDLING OF SMOKE.

BY

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PERSONAL AND EXPLANATORY.

THE literature of smelting lead, gold, and silver ores with blast-furnaces is limited in volume. It has mainly emanated from students of Chemistry and Metallurgy who have not enjoyed the advantage of personal contact with the practical problems confronting the lead-smelter. I recall no publication of this character that is not recognized as a distinct aid to research in the almost boundless realm of the twin sciences. But their dependable usefulness is limited to the didactic. The chasm separating theoretical from practical results is sufficiently wide and deep to engulf unmeasured capital and vaultless ambition. The theory of Smelting is a prerequisite to successful practice, but its mastery does not complete the equipment. Experience in applying the principles of Chemistry

and Metallurgy to the hard problems of Smelting activity, under widely varying conditions and environment, is a no less important factor in achieving financial as well as theoretical triumphs over Nature. My initial endeavors were almost coincident with the birth of the industry of lead-smelting in the United States. Others had preceded me in the direction and management of smelting plants, but their contemporaneous experiences and observations were to me a sealed book. The knowledge acquired during a period covering two decades came through direct contact with the sterner realities of smelting life, and by means of tests and experiments incident to the handling of a wide range of mineral substances, by all the manifold processes and devices discovered from time to time. The net results of twenty years' operations easily might have been multiplied had the unrecorded experiences of contemporaries been accessible to me. To the younger men of the period, on whose shoulders rest the responsibility for maintaining steady progress in the science of Metallurgy, I cheerfully dedicate this volume, in the confident hope that it may be serviceable in the solution of problems along the

broader lines, and useful as well in many of the lesser details.

Without attempt at literary embellishment, and with perhaps censurable disregard for niceties of diction, I go directly to the pith of the subject. My endeavor shall be to give novelty to that which was old, condensation to that which was diffuse, perspicacity to that which was obscure, and accuracy to that which was recondite. I shall attempt truthfully to relate what *has been* and *is*, humbly asking the considerate reader

"Gently to hear, kindly to judge."

THE AUTHOR.

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BLAST-FURNACE.

In most of the books written on lead-smelting considerable attention has been given to the subject of smelting lead ores in reverberatory furnaces. Such practice seems to be better adapted to foreign countries, where conditions obtain that are not to be encountered in the United States.

The amount of ore treated daily in a reverberatory furnace is quite small, compared with the tonnage of the larger lead-smelters. It is only fair to say, however, that great progress has been made in increasing the tonnage in reverberatory furnaces treating copper ores, especially noticeable at Butte City, Anaconda, and Great Falls, Montana.

A hitherto serious objection to the use of reverberatory furnaces was the great metallurgical losses of lead, gold, and silver through the smoke. It has

been demonstrated that the bag-house will save the values contained in the fume produced in the smelting of all kinds of lead, gold, and silver ores with blast-furnaces. Relatively it would be an easy matter to save the values in the fume from all classes of reverberatory furnaces smelting lead ores. It is important not to confound reverberatory *roasting* furnaces with reverberatory *fusing* furnaces, for it is known, from experiments coming under the author's personal supervision, that the fume from reverberatory roasting furnaces cannot be saved by the bag-house.

It was accidentally discovered, however, that the fume arising in the roasting of ores, at a point 12 feet from the bridge-wall, could be saved by cloth filtration; back of this distance it quickly destroyed the cloth, owing to the presence of sulphuric acid. The chief reason why the reverberatory cannot compete with blast furnaces lies in the fact that it is nearly impossible to produce a clean slag, that is to say, a slag that does not run high in lead; and hitherto difficulty has been experienced in the production of a uniform slag, possessing sufficient liquidity to enable the matte globules to completely settle to the

bottom. The reverberatory fusing furnace may be used in the future for lead-smelting, particularly when the rapidly increasing size, the use of gaseous fuel, magnesite brick, and other material known to withstand the corrosive action of oxide of lead are taken into consideration. A large reverberatory furnace may become practicable, by following the lines of large tilting furnaces, and using gaseous fuel for roasting, fusing, and final reducing action (decomposing silicate of lead), and adding the proper fluxes to obtain a fluid slag that will not hold matte globules in suspension. The slag so formed to remain quiescent for a period sufficiently long to dispense with all kinds of forehearths or other separately removed slag-settling pots, basins, receptacles, or furnaces.

It is believed that metallurgists educated in Germany first introduced, and successfully operated, the blast-furnace for smelting ores in use in the western portion of the United States. These furnaces were small, and some of the practices would now be considered rude and wasteful; still the right kind of furnace was indicated, and it was only a matter of time until marked improvements should take place,

both in the style of construction and the method of operating.

Before the publication of Dr. Hofman's book on lead-smelting the literature on this subject was widely scattered, and generally was of a fragmentary character. The aim here is to give the personal experience of the writer in operating lead blast-furnaces. Reference will be had chiefly to the period from 1880 to 1900, devoting major attention to the methods and practices in vogue at the close of the year last named.

By inspecting the drawings in Prof. Emmon's book on "The Geology of Leadville," it will be noticed that the furnaces were small and inexpensive; they indeed were primitive affairs when compared with the furnaces in use in the beginning of 1900.

Assuming the furnace has been charged and started, the first thing to do will be to recalculate the charge, by checking the figures, to see that no mistake has been made. At first a rather pure slag should be made, that is to say, one where the total percentage of silica, iron, manganese, and lime are high, aiming to prevent too much zinc, alumina, and baryta from entering the slag, which should have

about this composition: silica, 32 per cent.; iron and manganese, 26 per cent.; alkaline earths, 20 per cent. Be very careful at first to keep the percentage of iron plus manganese near the figures 25 to 27 per cent.; make a rather basic and fluid slag for a few shifts, even if it be the practice to form a high silicious slag. It should not only be thin, but very hot. Do not try to be too economical in fuel at first; let the minimum quantity of fuel be 15 to 17 per cent. That there may be no mistake on this necessary point, a typical charge is here given: 1000 pounds ore mixture (including iron fluxes); 70 pounds limestone; 330 pounds slag. For this use for the first shift, at least 240 pounds, or 17.14 per cent. of coke. When the slag continues to run hot and thin, then only should the percentage of fuel be reduced. Use 230 pounds the second day, 220 pounds the third, and then cautiously reduce the fuel to 200 pounds. This should be sufficient for smelting a mixed charge of 1400 pounds. Keep the pounds of coke as nearly a fixed quantity as possible, and gradually use ore fluctuating to 1200 pounds, but, generally speaking, 1100 pounds of ore mixture, including the iron flux, is preferable. At first do not use too much lime, since

excess will produce a somewhat curdy and brittle slag; it will not show a long fine thread when a bar is dipped into the fluid, and is more likely to cool quickly, producing a heavy thick shell.

The first slag produced should contain from 31 to 32 per cent. silica. After running the furnace two or three days, the silica may be increased above 32 per cent., gradually lessening the combined percentages of iron and manganese below 27 and even to 23 per cent.; but rarely is it best to use less than 24 per cent., and the alkaline earths will be increased from 18 to 24 per cent. Cut the iron as the lime is increased, and usually raise the silica.

Blow in the furnace at 8.00 A.M. After it has been producing slag one hour, take an assay and a laboratory sample. Immerse the cold end of a bar four inches into the slag; in half a minute remove and plunge it, with adhering slag, into a bucket of water; the shell will soon cool, and can easily be dislodged by gently tapping the bar. Take the assay and laboratory sample in the same manner. This slag is soluble in mineral acids. Drain the slag samples closely, and transfer them to granite-lined cups; dry the samples in these cups over the hot slag, and take

them to the assay office, where they should be powdered and passed through a 100-mesh sieve. After a thorough mixing, give them to the assayer and the chemist. Use a special bucking-board and muller, and never allow any ore or furnace products to be used on the grinding-plates. The slags should be assayed for silver and lead. The first trial slag that comes to the laboratory has the following determinations made: silica, iron, manganese, and lime. These should be completed in one hour from the time the sample reaches the chemist.

The first trial assay and preliminary analysis will enable the metallurgist with confidence to proceed with the smelting. They will indicate if the main slag constituents are properly porportioned and suggest changes to produce a different result. If the lime be too high or too low, he will make the changes quickly, since, if the lime be too low, the slags are almost certain to run high in lead. Sometimes in starting a furnace the lead in the slag will be high because cold; this may happen from running the water-jackets too cold, or from insufficiency of fuel.

As soon as the charge has settled it is important to take other slag samples, testing it for silver

and lead by fire methods. The wet method is used rapidly to ascertain the percentages: silica, iron, manganese, lime, baryta, magnesia, zinc, and alumina. Usually the magnesia is the last determination, but the percentage can readily be approximated. Occasionally it is advisable to determine the percentages of other components of the slag, as copper, sulphur, lead, phosphorus, potash, soda, and gold. It is best to keep a sharp watch on the copper entering the slag, and so likewise is it an excellent practice to save the buttons from the lead assays, and every week or month cupel them, to ascertain the gold losses.

Keep ample water on the jackets, in blowing in the furnace. See that the crucible is quite hot. Do not rod the furnace until it makes hot, thin slag.

It requires some time to thoroughly heat the furnace walls. When the slag runs high in lead, it is often caused by insufficiency of fuel at the start, frequently by using too little lime; therefore keep the fuel well advanced until the lead losses are under control.

There are only a few things that stop blast-furnaces smelting on lead ores, and these stoppages

usually are termed "freezing up a furnace"; as a matter of fact, the stoppages are not often due to shortage of fuel. Freezing up a furnace generally is traceable to the use either of too much alkaline earths or zinc. Cold silicious slags run slowly and show black streaked lines. They flow sluggishly through the slag-spout, and begin to run slow on tapping, but run fairly well after being well started; particularly is this true when much matte is present. These slags never freeze up a furnace, and a little more fuel, a trifle more iron or lime will generally correct the error quickly. It is the slags that tap badly—the breast being hard and thick and requiring the use of a small rod to start it—that is dangerous. They often pile in the spout, filling the slag-trough quickly. They are extremely brittle, the slag adheres to the rod and soon becomes thick both on the tapping-bars and the coke-rods. The shells on such slags are thick and brittle. This kind of slag shows, when liquid, a wrinkly, fish-scale appearance. It is a slag that freezes up a furnace, and gives rise to the chief trouble in lead-smelting.

The way to correct it, as if by magic, is to sharply lessen the lime. Sometimes, if heavy zinc ore is re-

placed in part by a pure silicious ore, in addition to the sharp reduction of lime, it will be advantageous. In an emergency add extra fuel and slag. The use of the so-called "half-charges" is productive of good results, but they are subject to much abuse. The half-charges follow: one half the ore mixtures, including iron flux; one half the limestone; but use the full quantity of slag and fuel called for by the regular charge. It is important frequently to note every little phenomenon about the slag; its physical properties should be studied at the instant of tapping. If, for example, it starts lazily and in a large, oily stream, there is denoted a safe silicious slag.

There is one kind of silicious slag that must be specially watched; it is the slag hitherto described as "silicious limy slag." This runs high in silica, low in iron, and high in lime or alkaline bases; and only when there is much zinc present does it give much trouble. This slag fuses at a much higher temperature, hence the fuel must at once be increased when such slag is observed.

It is not practicable to wait for the analysis of this slag; when the tuyeres begin to clog up, the breast of the furnace gets so hard it is difficult to start the

slag. It is also brittle, but it does not clog the spout. It soon causes the jackets to cool by the formation of a heavy crust; the tuyeres darken quickly. It is severe on the matte-boxes, causing them frequently to be sent out on the slag-dump. The silicious limy slag can be run a short time when absolutely necessary. but the "low-silica high-lime slag" is always more dangerous, and the instant it becomes curdy the furnace is in imminent danger of freezing up.

It is of the utmost importance to learn to distinguish at a glance between various kinds of slags. When the lime is largely replaced by baryta and magnesia the task is not simple. When the silica is low and much magnesia and zinc are present the slag becomes curdy and chills with alarming rapidity. The fluid slag in the pot becomes dark-colored and stony, the shell splits on cooling, and a light fire-line is seen. With slag of this character, cut the lime quickly and sharply, and in emergency cases do not be afraid of taking off too much. A drop of 50 to 100 pounds frequently is made, and in times of great danger drop the furnace down, and give one or two charges of coke extra; then give one slag charge, with the usual coke allowance, thus: 1200

pounds slag, 200 pounds coke. Continue with the regular charge, with nearly all the lime left off, until the furnace indicates decided improvement. By bold and quick action the furnace readily responds to treatment.

The furnace again running smoothly, the slag, although apparently good, begins to increase in lead; the remedy is in the addition of more lime cautiously applied. The chief difficulties in smelting are too much lime and zinc and too little silica. None of the following substances are likely to give more than temporary embarrassments when smelting large quantities: silica, iron, manganese, baryta, lime, magnesia, alumina, potash, or soda.

Blast-furnaces smelting identically the same ore mixture—the same size furnace, same blast, same fuel, and the same charge in every respect—often act very different. At one time, in running eleven blast-furnaces, all virtually the same size and on the same charge, the blast and all other conditions being as nearly alike as possible, the composition of the slag produced by each furnace was different. It is not uncommon for one or two furnaces out of a dozen in operation under precisely similar condi-

tions to yield a bad product. A remedy for the evil is not difficult to find.

Take a general laboratory sample on the furnaces working well, and a sample on the badly working furnace, and have these two samples analyzed for silica, iron, manganese, lime, baryta, and zinc. The difference in analysis will be apparent. Make a special charge for the abnormally working furnace, so the slag from it will correspond with that of the others. Figures may not be wholly relied on. Sometimes a whole line of furnaces will run uniformly and satisfactorily on a given charge for two or three days; the tonnage will be good; the slag will run low in both lead and silver, but suddenly one or more of the furnaces will begin to make a bad record. In such a contingency take a bar and break the crust on a pot of slag; see if it breaks short or with a string; note the surface of the cooled slag, whether smooth or rough; if warty, note if there be a tendency to form little volcanoes; turn the shell over, and if it be rough and shows unfused particles of ore, it denotes that fine ore is sifting down; if the slag is curdy, short, brittle, cools easily, and does not show a string on the bar, then cut the lime

quickly—30 to 100 pounds on dangerous occasions. Generally a cut of 30 to 40 pounds, with an addition of 50 pounds of pure silicious ore, will constitute a judicious change. If the slag taps hard, shows a decided string on the bar, then increase the fuel, cut the lime heavily, and raise the iron.

Baryta makes itself manifest by the deep green flame observed when the slag and matte are flowing, by the stony appearance of the slag, the large amount of matte, and in other ways.

MAGNESIA present in large quantities calls for a sharp increase in the silica; this base is 1.4 times stronger than lime; this slag is also apt to be stony in appearance; it fuses hard, and more fuel is required. Cut lime and alkaline earths when greatly troubled by magnesia, and increase the silica.

ALUMINA in excess acts singularly; it makes the slag stringy, wavy, decidedly viscous. It slows down a furnace a great deal when much is present; it is quite accommodating, acting sometimes like a base, but generally as an acid. With the high furnaces of the iron-smelters, the temperatures there obtainable, and the high percentages of silica, it is quite certain that alumina plays the rôle of a base;

but with the lower temperature of lead and copper smelters, and the basic character of the slag, alumina most generally acts as an acid.

The eye can be trained to indicate with almost mathematical accuracy whether a certain slag is good for the water-jackets or the tuyeres, whether it will be likely to chill the matte-boxes quickly and necessitate their removal.

A careful calculation of the charges, and two complete analyses of the ore-beds, is advised. Where they are high and unevenly prepared, three analyses should be made. Analyze also all the side ores as well as the iron flux. Take from the limestone and coke cars small samples, in order to accumulate samples each week. The roasted ore should be analyzed for silica, iron, sulphur, and zinc at least twice a week, or oftener when the roasting-furnace charge is changed. In short, the metallurgist should be furnished with complete chemical information at all times. Every material that goes to make up the charge—the lime, coke, side ores, iron flux—should be carefully scrutinized. The smaller the furnace the greater will these troubles be. It is easier to run a large than a small furnace. The

mechanical condition of the interior of the furnace, and the danger from an improper slag, are to be considered. Past studies on and experiences with the physical properties of slag must guide, since a solitary crystal may indicate the remedy. In making emergency changes on blast-furnaces there usually is an accumulation of certain substances that may produce disturbance. In such cases an aperient charge, such as a physician would give a patient, is the remedy. The best charges are those low in lime, zinc, and alumina, but with the silica and iron sharply advanced. Be careful not to have so much silica as to produce a slow-running slag—30 to 33 per cent. is about the right point; use from 26 to 28 per cent. iron. When the desired result has been produced, gradually lessen the percentage of iron.

Ordinarily it is best not to allow the silica to drop below 30 per cent., and rarely should it exceed 35 per cent. There are, however, exceptions to this. For example, 28 per cent. silica was quite ample when smelting much aluminous ores, and had all the properties and appearance of a high silicious slag. Successful runs have also been made on a

slag containing 43 per cent. silica, but this may be regarded as exceptional. The range of alkaline earths in slag is very wide and many things occasion it. With high silica use high lime. With high iron or zinc lessen the lime. There is not a more difficult problem in lead-smelting than the judicious handling of lime in the presence of much zinc. The lime is sometimes cut too sharply, causing the lead temporarily to increase in the slag. There is a constant conflict between the proper running of the furnace and high lead in the slag. High lime makes the slag low in lead. As the lead gradually increases it must be fought persistently with lime. The judicious handling of zinc ores on blast-furnaces is of major importance. For ordinary smelting the amount of zinc will vary in the slag from 4 to 6 per cent.

The presence of zinc is denoted by the flame, the dense white smoke, the volcanic explosions on a cooling pot of slag, and also by the deep yellow coat (yellow while hot, and white on cooling), having the appearance of sulphur.

In smelting much zinc ores the type slag—30 per cent. silica, 40 per cent. (FeO + MnO) and 20 per

cent. ($CaO + BaO + MgO$)—has been shown to be valuable.

It is a matter of the greatest importance to constantly observe the slag, to see that the lime is not too high, when smelting ores containing much zinc. There is a constant struggle to keep the furnace in good order, to produce a slag that can easily be handled, and at the same time keep the lead out of the slag. It is necessary constantly to approach the "danger line." A slag that is good for the furnaces is not necessarily a desirable slag so far as silver and lead losses are concerned. It requires more fuel to handle a slag with a high percentage of zinc oxide.

Some metallurgists have observed that there is a tendency for slag to contain from 2 to 3 per cent. lead when the zinc is unusually high. Under such circumstances the fuel and lime should be appreciably increased. While 13 to 14 per cent. fuel usually is quite sufficient for lead smelting, yet when the slag contains much more than 6 per cent. zinc a higher temperature is advised, and this should be gradually increased as the zinc increases. When the conditions are such as to compel the production of a high

silica and zinc slag, the utmost attention should be given to the fuel subject. Iron lessens the fusing-point of slags, and for this reason high zinc slags vary widely as to fusing-point. Where slags run high in zinc and are basic from iron, it is far more fusible than when basic from lime.

The reducing power of lead-furnaces varies widely, and it is best not to have the water-jackets wholly perpendicular. The walls should slope outwardly, from all sides above the jackets, to the feeding-doors. The reduction of these furnaces is dependent to a considerable extent not only on this flare, but also on the widening of the furnace toward the top; with zinc ores these lines are often quickly distorted.

Wall accretions and rapid driving are conducive to the entrance of lead in the slag. A sudden chilling of the furnace, due to insufficient fuel, to the use of too much lime, or coating the ore or fuel with zinc oxide, will produce sudden abnormal changes: the jackets will nearly cease to take water; they become quite cold; the tuyeres darken, and a solid mass will form in front, which is with difficulty penetrated by a bar, and entirely obstructs the passage of air into the furnace. The remedy is

more fuel and iron and less lime. When possible reduce the percentage of zinc, using some coarse, clean, silicious ore.

There are many times when scrap-iron has a marked effect in lessening the amount of lead in the slag and matte; but experience has shown that it is not well to use scrap-iron when the percentage of zinc is high in the slag. The decomposition of sulphide of zinc by iron produces metallic zinc, which burns with a blue-white flame in the upper part of the furnace to an oxide, and coats the furnace walls, as well as the ore and a part of the fuel, causing it to burn irregularly, if at all. Coke, covered with zinc oxide, causes the furnace to hang and work irregularly in the jackets. Under such conditions use a hot silicious slag.

During all the conceivable irregularities and abnormal workings of blast-furnaces, the Iles Slag-settler will be a most valuable adjunct, preventing heavy losses of matte.

The metallurgist should have printed sheets on which he can quickly figure a blast-furnace charge.

A table of the atomic weights of metals should be convenient.

In the analysis of slags rapid methods represent great savings. Accuracy is not necessarily sacrificed by haste.

The following conversion table will be found a labor-saver: $Fe \times 1.29 = FeO$; $Fe \times 1.43 = Fe_2O_3$; $Mn \times 1.29 = MnO$; $Zn \times 1.25 = ZnO$; $Cu \times 1.25 = CuO$; $Pb \times 1.08 = PbO$; $S \times 2.50 = SO_3$; $As \times 1.32 = As_2O_3$; $Sb \times 1.27 = Sb_2O_4$; $S \times 7.47 = PbS$.

Potash and soda cause the slag to run high in lead. With these alkalies good results may be obtained by using 17 to 22 per cent. alkaline earths, producing a very hot slag. With the oxide and sulphide of lead, they form several definite chemical salts, which doubtless facilitate the passage of lead into the slag. Slag containing considerable potash and soda is extremely tough, and large flat sheets can be raised with a bar when the slag is melted.

It sometimes happens that the matte formed by the same metallurgist at one place is low in lead and at another high in lead. It is not easy to explain such phenomena.

Slag of insufficient temperature containing much over 33 per cent. silica is quite apt to produce matte high in lead. A sharp increase of iron, lime, or fuel

often will materially decrease the lead. Where the matte continues high in lead an immediate lowering may be effected by the use of considerable scrap-iron. The general statement may be made that slag running 25 to 28 per cent. metallic iron is best suited for the formation of matte low in lead; also that a cold slag usually accompanies matte high in lead. Copper has a marked influence on the lead that enters mattes. Up to 50 per cent. copper the lead generally increases with the increase of copper, and when the matte runs 40 per cent. the lead is apt to be 20 per cent. Often the lead in the matte will run low for a long period—say five months—and then rapidly increase without apparent cause. At such times it is best to make frequent copper concentrations, sell the copper matte, or Bessemerize it, saving the fume in a bag-house.

The usual blast-furnace method for concentrating the matte would be to proceed thus: When they contain 15 per cent. copper, roast them alone and slightly cinter; draw the product out of the furnace into pots; hammer the surface hard with a heavy disc; make three holes through the centre of the matte; break the cooled product into large pieces

and smelt it with pure silicious ore, as free from sulphur, silver, lead, and gold as possible; the higher the silica percentage in the ore the better. For this concentration run the silica 35 to 42 per cent. Generally it is difficult during the concentration to make a slag that will run less than 1 to 2 per cent. lead. Constantly increase the lime on the charge until the lead in the slag is under control. Instead of using all limestone substitute on the charge 100 to 150 pounds alkaline earth ore. This will have a marked tendency to keep the lead out of the slag. The matte should be roasted to 4 per cent. sulphur. All forms of sulphur should be kept from the charge, even exercising care to use slag as free from it as possible. The charge should be made only of roasted matte, silicious ore, alkaline earth ore, limestone, and a small quantity of slag and coke. The charge is wholly dependent on the analysis of the various constituents, yet it may be indicated thus: 600 pounds roasted matte; 400 pounds high silicious ore; 100 pounds alkaline earth ore; 200 pounds limestone; 170 pounds slag; 200 pounds coke. The fuel percentage is 13.33, based on the ores and fluxes. Use a furnace with no wall accretions, and blow it in

according to the usual manner; run it on a rather high lead charge for a few shifts, and then put it on a slag charge, until all the ore is washed out. The start is made with a clean lead well; a clean crucible, no crucible crust, and the side walls practically free from sulphur. The blast does not have to be high, because the roasted matte, silicious ore, limestone, and slag are coarse, and generally nothing but the alkaline ores are fine. For this concentration 25 to 30 ounces blast will suffice. If much over 40 per cent. silica is found in the slag, there is generally some difficulty with the tuyeres; but this trouble is quickly overcome by running a 35 per cent. silica slag for a few shifts. The slag may contain as high as 45 per cent. silica. For steady work a range of 35 to 37 per cent. is better. The heat from this slag and matte is very great, and difficulty is experienced in keeping the matte-box from burning. Line it with silicious fire-brick; rowlock the bottom; have a 4½-inch fire-brick lining on the sides, and completely arch the top with fire-brick. As a precautionary measure, pass this slag through the slag-settlers. The matte is too valuable to lose even a fraction of it. The slag, owing to its

purity, quickly heats up the settlers, and cuts out the crust. The tap-holes of the settlers are opened easily. The production of any grade of matte desired is possible by regulating the sulphur. When it is run to 50 per cent. copper, "bottoms" are formed. These are returned to the blast-furnaces, causing a certain amount of copper to be reworked. The lead from the ores will extract much of the gold and silver from the bottoms. A better price is obtained than where silver and gold is sold in the matte. The lead in 50 per cent. copper mattes will usually run 15 to 20 per cent.; they do not command a ready sale. Purchasers are willing to pay for the silver, gold, and copper, but do not want to pay for the lead, claiming it is a nuisance, and they prefer not to have it, even when it costs nothing. They also claim the lead cuts the furnace walls, increases the losses by volatilization, and locks up much value in the bottom of the furnaces. These mattes fuse easily, but are roasted with difficulty, and when an effort is made to volatilize the lead the results are far from satisfactory. As the matte fuses easily, it is not subjected to the oxidizing influence of the blast for a sufficient length of time. The 50 per cent. copper

mattes may possibly be Bessemerized in a barrel converter, lined with magnesite brick. The lead would rapidly corrode a quartz lining. A high aluminous brick would last reasonably well in a barrel converter. In following this plan a bag-house should be used to collect the valuable fume. The fume produced by bessemerizing mattes, whether of lead, copper, nickel, cobalt, or iron, should be saved by filtration through cloth. There is no union between the oxide of lead and magnesia; hence the magnesite bricks would be ideal, the only objection being their cost, and the increased life of the lining might justify that. Little or no silica would be required in the barrel converter, as it is believed the lead oxide would mechanically carry the iron oxide. As a matter of fact, most of the lead would assume the fume state.

By the concentration of copper mattes on a bath of metallic lead some gold will pass into the lead; a considerable quantity will enter the copper bottoms, but this is desirable; 65 per cent. of the silver is extracted from the mattes by blast-furnace smelting. If the copper concentration is run long enough, the lead-well will close; but it is not likely to do so for a

month. Copper concentrations should be made frequently, otherwise the matte becomes highly charged with arsenic, the copper requiring handling several times. It has been noticed after a long copper concentration the lead in the matte will be low, and the bullion will be quite soft and more easily refined.

In roasting mattes that have been re-worked for a long time there will be driven off much arsenious oxide, which settles around the stacks and often is noticeable at considerable distance. On the top of matte cones there will also be observed crystals of arsenious oxide. Some metallurgists prefer to roast the matte, leaving 5 to 7 per cent. sulphur, and then smelt the roasted product with silicious ores, running the copper in the matte only to 25 to 35 per cent., roasting this product and bringing it up to blister copper in reverberatory furnaces. This process is satisfactory in cases where there is a sufficient quantity of copper matte and copper ores to keep the reverberatory furnace running constantly. Much trouble, however, may be experienced in making the furnace bottoms stand when the furnaces are run irregularly; the unused bottoms, however well made,

will swell up and undergo changes similar to the slaking of lime. To attempt to use the bottoms after long standing would prove a loss of time and money. The irregular use of the furnaces is also bad, since it takes much skill, patience, and training to secure and discipline a crew of men for spasmodic copper concentrations.

The Bartlett cupola is a small furnace with a sloping bottom. The air passes into it through ¼-inch slots, 8 inches long, and the height of the charge is only 18 inches. One of this type was successfully installed at a lead-smeltery for burning out the lead in matte, and although the furnace ran fairly well it was not wholly satisfactory. Some lead was burnt out, but a great deal was left in the matte; usually the matte was brought down to 7 to 10 per cent. lead, and for a few hours ran 4 to 5 per cent. lead. The quantity handled was small, and the work imperfect and irregular. The slag produced by this furnace is generally quite free from lead and silver. It was supposed to be susceptible of making a high silicious slag, but in every instance the silica was found to run only 28 to 30 per cent., and the silicious appearance was

due to alumina. It was found impossible to make a silicious slag continuously. Resort finally was had to a slag of this chemical composition: 29 to 31 per cent. silica; 26 to 30 per cent. Fe+Mn; 10 to 17 per cent. alkaline earths. The fuel percentage is supposed to be small, ranging from 8 to 10 per cent., but as the furnace frequently freezes and needs "barring off" and "dropping down," the operators control the quantity of fuel. Under such circumstances precise calculations cannot be made. This furnace originally was devised for the treatment of zinc ores, for which purpose it doubtless serves well. It cannot be commended for a modern lead-smeltery.

The details of some slag Bessemerizing experiments made in 1893 are worth traversing. A 1-inch rubber pipe was attached to the peephole of a tuyere, connected at the other end with a 4-foot length of iron pipe. The end was inserted in a pot of slag, and the air passing through it produced a dense white fume. A year later a second experiment was made. A slag-pot was run close to the furnace, a bell-shaped hood placed immediately over it, and the hood connected with a No. 1 Sturtevant

fan by a 6-inch pipe which connected a square sheet-iron box to the top of which a 30-foot muslin bag was attached. The air from a compressor was blown into the molten slag in 41 small pots, and there was collected 66 ounces fume, fairly white in color, having a light grayish tint. The fume was compact and heavy. Samples of the slag were taken before and after this experiment; before treating it contained 1.5 per cent. sulphur, existing in the slag mainly as matte. The sulphur in the slag is somewhat a criterion of the quantity of matte. This does not, however, hold good with reference to other slags produced by dissimilar methods.

The quantity of zinc lost was very small, the silver remained the same, and the lead loss was $\frac{1}{10}$ per cent. The fume collected was small compared with the slag treated, and showed that 1 ton of slag will produce .84 pound of fume, with a commercial value not to exceed 2 cents a pound. The fume was carefully analyzed, giving this result: 16.1 per cent. SO_3; 6.9 per cent. total sulphur; 7.9 per cent. zinc; a trace of copper; and 2.5 ounces silver, a trace of gold. The wet determination on lead gave 64 per cent. The chemical

elements of the fume are probably in combination as follows: 1.51 per cent. sulphide of zinc; 8.58 oxide of zinc; 60.94 sulphate of lead; 24.07 oxide of lead; a total of 95.13 per cent. The balance possibly is potash, soda, arsenic, antimony, etc. In the second experiment the same apparatus was used and the operation conducted precisely in the same manner, except that the slag was covered with coke. Nine slag-pots were operated on; there was collected 14 ounces of fume. One ton of slag produced 12.4 ounces of fume. The slag, before blowing, showed 4.1 per cent. zinc; after blowing, 4 per cent. The loss of .1 per cent. is, of course, small and indicates that even in the presence of carbonaceous matter *the zinc in the slag is not readily decomposable.* The fume showed by analysis practically the same results. It also showed a trace of gold, and the fume was somewhat darker than that obtained by experiment No. 1. The slight lowering of the percentage is due to the increased quantity of carbonaceous matter.

Experiment No. 3 was conducted exactly as were Nos. 1 and 2, except that charcoal was used as a covering. Seven slag-pots were treated, producing

11 ounces of fume. One ton of slag, before blowing, contained 5.4 per cent. zinc; after blowing, 5 per cent. Here is noticed the first appreciable loss in zinc, and even this is trivial compared with the total quantity of zinc present. No. 3 practically corresponded with Nos. 1 and 2, except that the color was darker, due to the presence of carbonaceous matter. This fume also showed a trace of gold.

Results may be summarized thus:

1. The zinc found in argentiferous lead slags exists essentially in the form of silicate, not decomposable by blowing air through it. Neither is it decomposable to any marked extent in the presence of carbonaceous matter. A small percentage of sulphide of zinc exists as such in the slag, both as a free sulphide of zinc and in chemical union with other sulphides.

2. The lead exists as metallic, silicate and sulphide of lead. There is also sulphide of lead in chemical union in the matte. There is some sulphate of lead, due to the continual oxidation of the sulphide. This constantly arises and produces the white fume so commonly noticed.

3. Silver exists in the slag as matte, held mechani-

cally. Many chemical experiments made by the author demonstrated that silver cannot exist in the form of silicate in slag.

DRAWINGS AND SPECIFICATIONS.

The original construction of the plant should generally be done by contract, except the furnaces; and subsequent alterations, additions, and improvements should also be by contract, as experience has shown that there is less interference with the running of the plant to its maximum efficiency. For such work the drawings and specifications should be full and complete.

The drawings should all be made to a scale, and the scales preferred are the following fractions: $\frac{1}{8}$, $\frac{1}{4}$, $\frac{1}{2}$, and 1 inch, representing 1 foot in each case. Every important dimension, depth of foundation, and everything, even to the size and length of bolts, should plainly be marked on the drawings, which should be so complete that even if the specifications do not state the length and size of bolts, they will be shown on the drawings.

Only skilled engineers and draughtsmen should be allowied to prepare the drawings. After they

are approved by the engineer a blue print should be taken and again carefully revised. Then it will be best to consider separately any suggestions from the mechanic, head carpenter, and head brick-mason of the works. Often excellent suggestions are obtained from them. It is of the utmost importance to interpret a mechanical drawing accurately and quickly. Frequently the success of the plant will depend largely on the accuracy of the drawings and the ability of the owners to understand every detail. To accomplish the best results have a mechanical and constructing engineer who is also a fine draughtsman at the works permanently. The superintendent and engineer should have the plans well digested before the tracing is made. Some convenient place should be provided the contractors in which to prepare their bids.

All large industrial plants should provide a vault for valuable drawings, and all blue-printing should be done at the works. The original tracings should be properly numbered, wrapped, and marked plainly for quick identification. At least two sets of blue prints should be mounted on cardboard; and where the drawings are large they should be made to fold

like a book. The original tracings should not be taken from the office; the contractors and builders should be furnished with mounted blue prints. Industrial specifications should be prepared with great care. There are certain set phrases and sentences which will be found common in this class of specifications. The subject-matter should be systematically arranged, and that part of the work which begins first should be mentioned first, and the heading of each subject should stand out in large, bold letters. Whenever the specifications cover a piece of work involving a large sum of money there should be an index of subjects; this will result in saving of time. It is best to print as much of the specifications as possible (the remainder typewritten), and at the close of every important paragraph there should be several blank lines.

As the specifications for a lead-smeltery are so different from other industrial plants, it has always been difficult to gain assistance from any book or paper.

The superintendent should be able quickly to judge the merits and demerits of the drawings, and when necessary be able to prepare all specifications

and contracts. It will be difficult to enumerate the multiplicity of small accessories to be built, either on the company's ground or at the foundry, machine-shop, or at other places.

In all cases, even to the building of boilers or engines, specifications should emanate from the industrial plant, although it be necessary to study the specifications of different establishments bearing on the subject. This, of course, takes time and some expense, but if the plant is to be a success financially there is no way to avoid it. For almost all construction work have a drawing, a specification, and a signed contract. The moment you depart from this trouble will arise. In the contract have a Time Clause and a Penalty Clause. The sum demanded in case the work is not finished on contract time need not be large, but it will have a very wholesome effect on the contractor.

At a lead-smeltery a few of the many things that must be built are, briefly: smoke-stacks, smoke-ducts, dust-chambers, both of iron and brick; brick, iron, and wood floors, platforms; cesspools and various closets and vaults; blast, roasting, fusing, and refining furnaces. All kinds of tools, implements, and

devices for the handling and transportation of material about the works should be made from drawings, accompanied by full and complete specifications.

It is difficult to prepare a good set of drawings and specifications by reason of insufficient data. In such cases the requirements should be carefully considered; go direct to experienced architects and engineers for suggestions. Frequently it will become necessary to visit other plants; this should be done when the subject is fresh in mind. Draw freely from all kinds of industrial plants in order to secure the most advanced ideas. Foreign countries should be visited where one has reason to believe the practice is better than in his own country. Industrial plants are great money-makers when properly managed. It is the modern up-to-date plants that yield the best returns.

This chapter is particularly pertinent to the lead-smelting industry; in a great many respects it will apply as well to all industrial plants.

When the drawings and specifications have been completed and it has been decided to go ahead with the construction, then letters should be addressed

to a number of contractors well versed in the distinct style of construction contemplated, inviting a call. The drawings should then be carefully explained, the specifications read to and by them, and a visit be made to the site and the place where the excess dirt is to be transferred. Facilities offered for railroad transportation, if any, and where they can attach their water-hose, should be fully explained. Give them to understand that they must not only furnish their own tools, but must take care of all their own property, and that while the company's men will not be allowed to interfere with them, their employees must not interfere with the company's men or property.

When the bids are all in they should be opened in the presence of the bidders. Each proposal should be read aloud, and the lowest bidder awarded the contract. The successful competitor now should be required to sign the original tracings and specifications, testifying by so doing that he understands them. Then the contract should be signed by both parties.

THE INNER LINES.

From 1879 to 1900 the inner lines of blast-furnaces used for smelting argentiferous lead ores varied widely. The size at the tuyeres varied from a circle 36 inches diameter to a rectangular section at the tuyeres of 60 inches width and 160 inches length. The bosh of the water-jackets varied from 6 to 12 inches. The cubical contents of the crucible varied from 8 to 77 feet (designated A). The cubical space enclosed by the water-jackets varied from 40 to 190 feet (designated B). The cubical space from the top of the water-jackets to the feed-floor level (considered the height of the charge) varied from 23 to 1223 feet (designated C). The size of the furnace is measured by the number of cubic feet it contains. Its capacity varied widely from 93 to 1500 cubic feet. The height of the water-jackets varied from 41 to 60 inches. The height, measured from the top of the jackets to the feed-floor, varied from 4 to 16 feet. The width of the furnace at the feed-door varied from 4 to 7 feet, and the length of the furnace at this place (all dimensions taken from the inside of the furnace) varied from

5 to 15 feet. The distance from the centre of the tuyeres to the feed-floor (or top of charge) varied from 10 to 20 feet. A few furnaces had a height of charge above the tuyeres of 20, 22, and 25 feet, and one 30 feet.

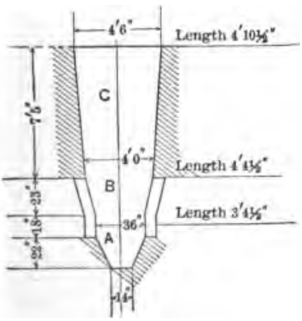
Observation and practical experience, covering a period of two decades, are comprehended in the annexed recommendations as to the dimensions of an ideal furnace:

Size at tuyeres.................	48×144 inches.
Bosh of jackets	10 inches.
Contents of crucible	77 cubic feet.
Contents of jackets............	180 cubic feet.
Contents from top of jackets to feed-floor....................	1243 feet.
Total capacity	1500 cubic feet.
Height of jackets	41 to 48 inches.
Top of jackets to feed-floor.....	13 feet.
Width at feed-floor.............	7 feet.
Length at feed-floor............	15 to 16 feet.
Height of charge (tuyeres to feed-door)......................	16 to 17 feet.

The concept of a furnace as above outlined would have good reduction, great speed, and long life; it

would make little flue-dust and the slags would be good, provided the charge was watched and correctly calculated. With a blast of 3 to 4 pounds, and tuyere openings $3\frac{1}{2}$ inches, there would be smelted daily (24 hours) 140 to 150 tons of ore, exclusive of fuel, lime, and slag, but inclusive of iron flux. The inner lines are more particularly indicated by Figs. 1, 2, 3, 4, 5, 6, 7, 8, and 9.

The reduction of different forms of lead blast-furnaces varies widely. The reasons are obscure.



A = 12.2
B = 41.5
C = 143.7

Total = 197.4 cu. ft.

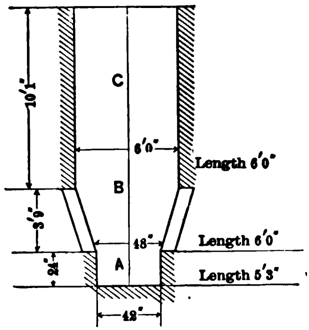
36" × $40\frac{1}{2}$" at tuyeres.
6" bosh.

Fig. 1.—Small Portable Furnace.

To ascertain this fact take two examples, Figs. 7 and 8. Fig. 7 shows the inner lines of a large tonnage producer, making slags quite free from silver and lead, with matte low in lead.

Fig. 8 shows a much larger furnace, but its ton-

nage is no larger than that of Fig. 7. The slags are high in silver and lead, and the matte is excessively high in lead. It is too narrow at the top, too high, too wide at the tuyeres, and has the bosh



A = 42.37
B = 112.5
C = 360

Total = 514.87 cu. ft.

48″ × 72″ at top of crucible.
12″ bosh.

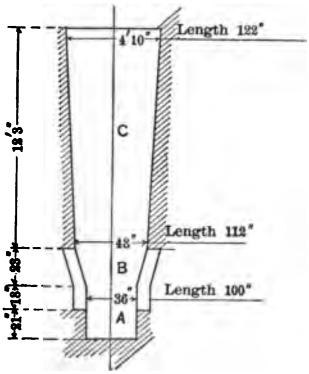
FIG. 2.—Straight Water-jackets.

in the brickwork, instead of a sharp 12-inch water-jacket bosh, and this furnace is faulty by having a 6-inch bosh in the jackets. It is impossible to maintain the bosh in the brickwork in the case of Fig. 8, but in Fig. 7 it is practicable.

Furnace Fig. 7 has proven an excellent type; Fig. 8 a great failure. One smelts as much as the other; indeed, Fig. 7 can be put inside of Fig. 8 with the inner lines appearing like Fig. 9.

Blast-furnaces and furnaces generally should be large; they may, however, be improperly built. The

matte on furnace Fig. 7 runs from 7 to 10 per cent. in lead; on furnace, Fig. 8, 20 to 40 per cent. As the copper increases in matte there is a marked tendency of the lead to increase, and for this reason concentrations should take place as soon as the matte reaches 15 to 20 per cent. copper, when using



A = 43.7
B = 93.4
C = 527.5

Total = 664.6 cu. ft.

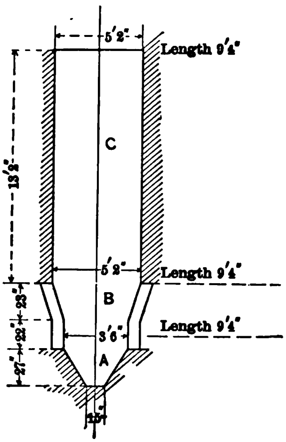
36" × 100" at tuyeres.
6" bosh.

FIG. 3.—Favorite Form from 1880 to 1890.

large furnaces and driving fast; but with small furnaces the copper in the matte when exceeding 10 to 12 per cent. occasions difficulties in the crucible and closes up the lead well. Slags high in silica should not be used when the matte is high in lead; a basic slag with the lime high is preferable.

The construction of a modern lead blast-furnace is practically the same as those smelting copper ores,

except the latter are shorter, and the distance from the centre of the tuyeres to the top of the charge is less than furnaces smelting lead ores. Copper furnaces have no crucible.



A = 50.9
B = 134.1
C = 634.9

Total = 819.9 cu. ft.

42" × 112" at tuyeres.
10" bosh.

FIG. 4.—Perpendicular Walls and Singular Crucible.

Furnaces ultimately may attain a capacity of 2500 to 3000 cubic feet, and doubtless follow the lines indicated by Von Rachette, Schinz, and Truran for smelting iron ores. The Rachette furnace was the guide the author used in designing the successful furnace called Figs. 6 and 7. An enlarged furnace mouth is strongly commended as it allows a more uniform descent of the charge; the furnace does not need to be so high, hence

the blast is less, and, what is more important, the ores become better heated, the gases are removed at a lower temperature, and a compound oxydizing and reducing furnace is secured.

The height of Rachette furnaces is somewhat greater than in furnaces subsequently built; still, if the ore is coarse, the height may not be excessive.

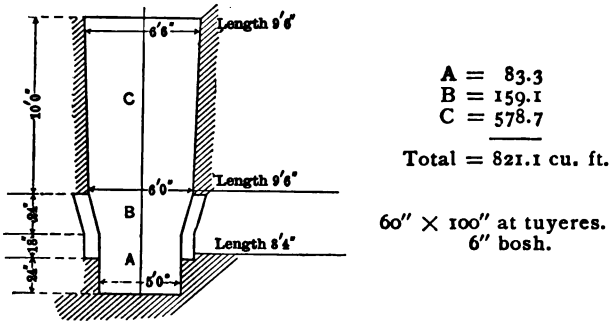
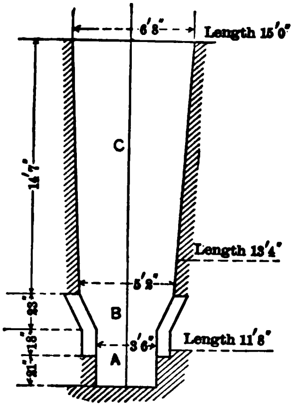


FIG. 5.—Wide and Short Furnace.

An interesting and valuable table giving the various dimensions of furnaces used at different times and places up to 1900 is appended. It gives the size at the tuyeres and water-jacket bosh, as well as the cubical contents of various divisions. In most instances the height of the water-jackets and the distance from the top of them to the top of the charge are given. In other cases the size of the

furnaces at the feed-floor appears, and generally the distance measured from the centre of the tuyeres to the feed-floor is given. Furnace indicated as Fig. 7 is believed to be the best of all the types. Lead and copper furnaces may be compared as follows:

	Lead.	Copper.
Area of horizontal section at tuyeres	140 × 42 40.8 sq. ft.	144 × 44 44.0 sq. ft.
Area of horizontal section at top of jackets	69.0 " "	64.0 " "
Area of horizontal section at feed-floor	100.0 " "	80.0 " "
Volume from feed-floor to top of jackets	1232.5 cu. ft.	522 cu. ft.
Volume from top of jackets to angle of bosh	105.2 " "	162 " "
Volume from angle of bosh to bottom of jackets	61.2 " "	110 " "
Total volume	1398.9 " "	794 " "
Bosh in jackets	10″	12″
Height of column to feed-floor	16′ 11″	11′ 6
Height of column actual in practice	16′ 0″	7′ 6″
Number and size of tuyere points	14—3⅛	16—4¾
Total combined area of tuyere points	67.38 sq. in.	283.52 sq. in.



A = 72.
B = 171.18
C = 1222.93

Total = 1466.11 cu. ft.

42" × 140" at tuyeres.
10" bosh.

FIG. 6 (side view Fig. 7).

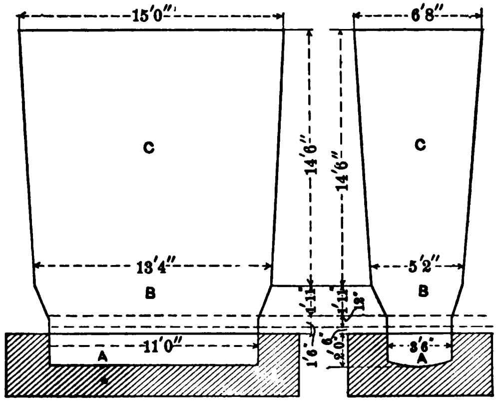


FIG. 7.

Lead Blast-furnaces. Inner Lines of.	Size at Tuyeres (Inches)	Bosh of Jackets (Inches)	Contents in Cubic Feet. A Crucible.	B In Jackets.	C Above Jackets.
Round furnace built for C. James	36	....	8	62	23
Small portable	36×40½	6	12.2	41.5	143.7
" "	36×40½	6	14.2	39.6	196.5
Early style furnace	36×40	6	12.6	39.3	209.6
Copper, Butte	36×96	10½	0.0	231.6	161.5
Built for C. J	33×72	....	28.0	62.0	182.0
Standard S. & R. Co.	33×84	....	21.0	76.0	275
Hot-blast pyritic furnace	30×140	6	0.0	152.5	246.4
No. 3 Germania Lead Works, Salt Lake City	41×76	8½	24.91	85.03	326.88
Portable Sultepec Mining Co., Mexico.	33×60	6	17.6	53.0	408.2
Yuba S. & R. Co	36×80	6	26.5	75.2	389.5
Furnace built at Leadville about 1880	33×84	6	24.7	72.9	402.5
No. 4 Germania, Salt Lake City	48×72	12	42.37	112.5	360.0
Built in 1892 for a small plant	33×80	6	22.1	76.3	443.4
Monterey, Mexico	36×100	....	28.0	99.0	401
Hot-blast pyritic	33×140	6	....	165.7	381.4
Grant Sm. No. 9 Phil. Mn. & Sm. Co., Pueblo. Pueblo Sm. & Ref. Co., 1883	36×80	6	32.9	75.2	439.9
Omaha Grant. Denver, 1883	36×80	6	32.4	76.1	437.3
1880 to 1886 furnace	36×80	6	32.9	75.2	454.8
San Juan & New York Sm	36×96	10	60	98.8	404.8
Colorado Sm. Co., Pueblo	36×90	10	54.37	94.34	423.2
Helena & Livingston	42×96	....	43	110	410
	36×100	6	27.6	93.2	451.5
Omaha & Grant Sm. & Ref. Co., 1884.	33×100	6	31.4	86.0	474.0
N. Pacific Sm. Co	36×120	....	33.0	118	422
	36×100	6	28.7	93.2	490.2
	36×100	6	28.7	102.5	582.3
Slag furnace	36×120	6	0	115.0	520
S. & R. Co. Furnace	36×100	6	50	93.4	510.4
Omaha & Grant, 1886–1892	36×100	6	43.7	93.4	527.5
Leadville Works	33×120	6	34.3	102.7	540.4
Furnace 142, Salt Lake City	42×84	9	62.9	106.2	520.62
Great Nat. Smelting Co., Mexico	42×120	....	40	137	545.0
Cia. Met. Mexicana	42×120	....	50	132	545
Nos. 9, 10, 11	30×120	6	43.75	99.25	664.8
Slag Furnace	36×100	6	52.5	93.2	665.7
	42×112	10	50.9	134.1	634.9
Pueblo	60×100	6	83.3	159.1	578.7
Zeehan Dundeas Sm. Co., Tasmania	40×136	6	35.64	140.82	714.01
El Paso Sm. Co	42×120	10	43.7	144	743.0
Mingo Co	36×120	6	46.3	122.2	780.2
Standard Sm. Co., 1888	42×120	10	77.0	165.32	875.0
Built in 1892	42×120	12	72	173	990
Philadelphia	60×120	....	60	190	1150
Fig. 7	42×140	10	72	171.18	1222.93

Total Cubic Feet.	Height of Jacket. (Inches)	Top of Jacket to Feed Floor. (Inches)	Dimensions at Feed Door. Width. (Inches)	Dimensions at Feed Door. Length. (Inches)	Centre of Tuyere to Feed Floor. (Inches)	Built by
93	....	...	....	....	....	Frazer & Chalmers
197.4	41	89	54	58	120	Hendrie & Meyer
250.3	41	129	50	54	160	Do.
261.5	41	118	60	....	....	Do.
393.1	......	51	57	108	60	Built at Butte
272.0	......	....	....	....	....	Frazer & Chalmers
372.0	....	....	....	....	....	Do.
398.9	55½	66	56	154	108	Colorado Iron Works
436.82	45	145	58	....	180	Built at Salt Lake City
478.8	41	186	54	80	216	Colorado Iron Works
491.2	41	126	54	80	132	Do.
500.1	41	139	54	104	168	
514.87	45	121	72	72	156	Salt Lake Foundry
541.8	42	167	52	99	198	Colorado Iron Works
528	......	....	....	.....	....	Frazer & Chalmers
547.1	55½	93	59	154	138	Colorado Iron Works
548.0	41	147	58	102	156	Colo. Iron Works, and others
545.8	41	147	58	102	177	Colorado Iron Works
562.9	41	144	62	106	156	Hendrie & Meyer
563.6	42	120	56	106	152	Do.
571.91	42	120	56	110	152	Pueblo Foundry
563	....	....	....	....	....	Frazer & Chalmers
572.3	41	126	58	122	156	
591.4	41	145	53	120	176	Colorado Iron Works
573	....	....	....	....	....	Frazer & Chalmers
612.1	41	138	58	122	200	
613.5	42	168	55	119	200	
635.0	41	156	48	120	187	Colorado Iron Works
653.8	41	174	48	120	204	Do.
664.6	41	147	58	122	158	Do.
677.4	41	144	51	138	175	Do.
689.72	43½	147	60	102	180	Salt Lake Foundry
722	....	....	....	....	....	Frazer & Chalmers
727	....	....	....	....	......	Do.
807.8	41	178	57	144	185	Colorado Iron Works
811.4	41	192	56	120	223	Do.
819.9	46	158	62	112	196	Kansas City Foundry
821.1	42	120	78	114	176	Pueblo Foundry
890.47	42	163	52	148	196	
930.7	46	173	62	120	209	
948.7	42	196	56	141	224	
1117.32	48	174	62	140	212	Hendrie & Meyer
1235	48	180	66	144	216	Do.
1400	....	....	.. .	....	....	Frazer & Chalmers
1466.11	41	175	80	180	192	Denver Eng. Wks. Co.

THE FOUNDATION.

Construct the blast-furnace building before starting the foundation for the furnaces and leave a proper opening in the feeding floor. From each of the corners of the rectangular openings plumb-lines should be dropped to lay off the furnace foundation. When the lines have been ascertained a hole should be dug, conforming to the lines, to the depth of three feet.

Hard red brick laid in gray lime mortar is preferred for the foundation. Grouting is done up to the last two or three courses, and the top is finished by laying several courses with greater care.

The furnace columns should extend into the ground 3 feet. The column foundations should start at the same level as the bottom of the large hole. In the construction of these use only hard paving brick and lay with full flush-joints, using gray lime mortar and the best Portland cement. In some cases, where the furnaces are built on wet, springy ground, use a concrete bed before starting the column foundations. In all cases it will be found good practice to place a large flat stone 8 to 10 inches thick at the bottom of the columns.

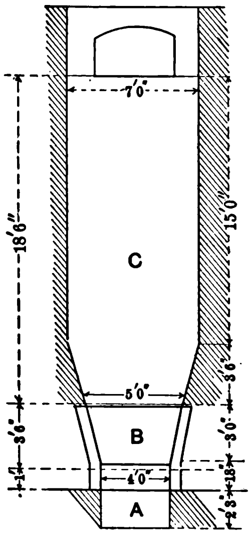
One of the chief reasons why the furnace columns should extend below the ground is that one can never tell what height will actually be required at the front of the furnace in the nature of fore-hearths.

The lead bullion that permeates the foundations of these furnaces becomes gradually richer in gold and silver. Where large, rapidly driven blast-furnaces are used, the lead in the crucible becomes quite hot, hence the absorption of the bullion into the foundation is greatly increased. When fire-brick is used the entire foundation becomes colored and very heavy, and on breaking the brick it is found threaded with bullion; but with common red brick, particularly when quite dense, comparatively little absorption takes place, and the mortar apparently has the property of becoming replaced. It is a waste of money to put in the foundation of these furnaces with fire-brick as there is no intense heat at that point; what is needed is firmness, compactness of structure, and some substance which has not the property of absorbing lead. As a matter of fact, red brick are far more porous than fire-brick, and one would naturally think red brick would absorb the greater amount of lead, but experience

proves quite the contrary. Blast-furnace foundations of molten slag have been constructed, but this has nothing to commend it; indeed, it is the worst possible foundation, although theoretically it would seem ideal. The insurmountable objection to a slag foundation consists in the fact that the bullion absorption is very large, and through the many thousand cracks in the slag a complete infiltration of molten lead, very rich in silver and gold, takes place.

Whenever it becomes necessary to remove the slag foundations, it will be found that there is presented a difficult mechanical problem, since it is hard to remove. Drilling holes, either by hand or with steam or compressed air, will be of little use, as it is impossible to blow up or dislodge any portion of the slag bottom of a size sufficiently large to pay for the trouble. It is quite immaterial whether black or giant powder is used.

The fact is recorded that the minimum absorption is from four to six thousand dollars, the maximum twenty thousand and over. It is a difficult matter to sample furnace bottoms; the best way is to keep the metallic portions entirely separate and then



A = 102.
B = 227.75
C = 1775.

Total = 2104.75 cu. ft.

48" × 136" at tuyeres.

FIG. 8.

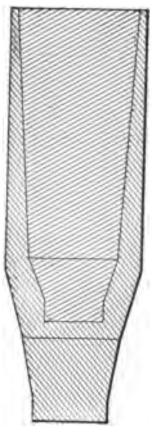


FIG. 9.

melt them down in a large cast-iron pot. The skimmings of the lead are rich and care must be taken not to get too many "metallics" into the portion to be crushed, otherwise the machinery is liable to break.

THE CRUCIBLE.

The bottom crucible plate is made of wrought iron with a thickness not less than half an inch. This plate in reality should be in one piece, as the trade sheets of this thickness of steel will not generally permit. It should have the fewest possible seams, and unusual care should be taken to make it water and air tight. The writer has serious doubts whether a joint can be made that will prevent the leakage of lead when the furnace has run for five or six months. He recommends the seams to be "double butt-strapped," with a bar half an inch thick 6 inches wide and riveted in place,—one bar at the bottom and another over the top,—using $\frac{5}{8}$-inch rivets.

Experience has shown the great desirability of covering the top of the brickwork of the crucible with cast iron plates. It is necessary to have some mode of securely fastening them down.

It is preferable to have all the side crucible plates made of cast iron at least 2½ inches thick throughout. These plates are reinforced with three rows of 75-pound rail; two rails are placed closely together, bent up at each end to catch the corner bolts. Between the two 75-pound rails run 1¾-inch bolts, using on the inside a heavy wrought-iron washer. The rail bolts should be placed at distances not greater than 18 inches.

The crucible plates should have an outward flange at the bottom in order to bolt them to the bottom sheet-iron plate. Leakages of lead take place frequently at the bottom edge of the crucible plates. Increased size of furnace imperatively demands stronger construction, and greater blast pressure means more tons of ore smelted each day and much greater production of bullion. On account of the length and the heat greater strength is required. If the crucible plates are 36 inches wide, a depth of 21 inches for the crucible has been secured, and this has been found to work well. Still, 25 inches is preferable, and in order to get this the plates may be 3 or 4 inches higher. These plates should not vary much from 36 to 42 inches in width. The bot-

tom ring course of the crucible should be built with great care. Use "key brick," and see that these, as well as all others in the crucible, are closely laid. Care should be taken not to have the fire-clay too thick.

The lead-well formally was put in the centre of the projecting side of the crucible, but modern practice has shown that it should be near the front. Formerly this channel was circular and had a diameter of only 2 to 3 inches, but now it is sometimes made 1 foot square; 6 or 8 inches in width is advised. On account of the frequent clogging of the lead-well on small furnaces much trouble, delay, and expense was occasioned. These difficulties were always intensified by the presence of copper in the matte, in excess of 10 per cent. Since the lead-channel has been so materially enlarged and the quantity of lead entering the crucible has been so largely increased by fast driving these troubles have almost entirely disappeared. The metallurgist can now run a furnace for a year, producing matte containing from 15 to 25 per cent. copper without the slightest difficulty and very frequently mattes running from 40 to 50 per cent. copper for

a period of two to four weeks have been produced; but at this high point the copper alloy should constantly be removed from the top and sides of the lead-well. Cold lead, both in the crucible and in the well, is now a thing of the past, except in exceedingly rare cases. There is not a single item in the various departments of lead-smelting that has so lessened the cares of the furnace attendants as this elimination of the lead-well troubles; nor has there been anything which has made this industry so safe and profitable. Whenever the lead-well channel was closed up the lead had to escape in some manner, and the only way was with the slag and matte.

The lead-well should be so constructed as to enable rods to be run straight down to the bottom of the crucible. There should be no bends or angles in the channel, and have from 8 to 10 inches of brick above the opening. The top of the lead-well is considerably above the level of the crucible. It really is a separate wrought-iron appendage, lined with brick, and is the last thing built about the crucible. For tapping the lead there is a 1½-inch hole in the lead-well plate near the bottom; the lead

flows through a small cast-iron spout. This band is anchored to prevent lead from oozing. The furnacemen prefer to have a large margin in the height of the lead-well, which they soon regulate to a great degree of nicety; of course, the height should vary according to the blast pressure—the higher the blast, the higher the lead should be carried in the lead-well. The bottom of the lead-well may be sprayed several times each shift with water to prevent lead leakage.

The covering plates at the top of the crucible are made of cast iron, in sections, and 1½ inches thick. These plates should fit snugly up to the side plates, and the nuts be counter sunk. They should have a slope from the water-jackets.

By the use of these plates the furnace is kept far neater and water is prevented from coming on the brickwork. The uncovered brick will swell, disarranging the water-jackets, presenting an uneven surface and a careless appearance. By allowing water frequently to run on the brick it causes it to disintegrate. It has often been noticed that fire-brick used in the crucibles of lead blast-furnaces will crumble when exposed for some time to the

atmosphere, and especially if water is occasionally sprinkled on them. This chemical change is due to an oxidation of the sulphides, producing sulphates of the various metals, particularly when the sulphur is combined with iron. This explains why unused crucibles crack.

Most excellent props for the jackets can be made from pieces of 30-pound railroad iron. When the furnaces were small and smelted from 40 to 50 tons of ore a day these props were unnecessary; now, by lengthening the furnace and by rapid smelting, something is needed to hold the jackets in alignment.

Hollow walls, to prevent radiation of heat from the crucible, are not recommended. The crucible has now a tendency to keep too hot,—so hot, indeed, that difficulty is experienced in constructing a basin that will hold the lead, particularly at the bottom. Provide a perforated water-pipe at the bottom of the crucible.

Make a heavy crucible and bind it securely. When the furnace is perfectly dry the crucible should be filled with lead in the manner described in the chapter on "Blowing-in."

THE WATER-JACKETS.

Water-jackets 4 to 6 feet in height should be made of cast-iron sections 6 inches thick and 20 wide. They should be set up one at a time on the cast-iron plates that cover the brickwork of the crucible; the jackets will snugly fit into the depression in the plates. When all except the two front jackets are bolted in place line them up. There should be a double set of 1¼-inch bolts near the bottom and top for holding them; there should also be a good-sized oblong hand-hole at the bottom. Use the same style yoke common to tubular boilers provided with rubber gaskets. When a furnace is blown out occasionally remove the accumulated mud and scale. The handholes sometimes leak for a time after starting, but rarely is trouble experienced after the third day. The tuyere hole is circular, tapering from the outside to inside. The tuyere hole, accurately turned, should be in the centre of the jacket, placed 12 inches from the bottom. The hole should be 3½ inches, measured inside; thickness of the tuyere point, ⅜ inch; is circular, with sufficient taper to prevent dislodge-

ment even at a high air pressure. A little above the top of the tuyere hole the jacket should flare outwardly with a batter of 10 inches.

The water-jacket should be free from sharp angles. The extreme top should have a distinct batter toward the inside. The only function of the neck is to provide a small quantity of water higher than the top of the jacket, preventing the top from burning. The water is discharged through a pipe at the neck into a copper water-trough, strongly reinforced on both sides. Connect all the jackets with each other by a rubber hose; in case the water in one jacket gets low it will be automatically fed from the others. The shortest, simplest, and lightest tuyere, with a turned cast-iron point fitting squarely into the hole in the jacket, will produce the best results. Use smooth jackets that fit closely along the entire side. A scale on the inside is quickly formed, and this effectually prevents air leakage between them. The loss of air is a matter of moment and should be carefully watched. It usually is traceable to the tuyeres. Jackets should be made of a soft, tough, fine-grained pig-iron, with a sufficient admixture of silicon pig-iron. They should be free

from warp, wind, blow-holes, sand-holes, or cracks. All jackets should be cleaned out and tested for leaks before leaving the foundry. The weight of each should be separately recorded when delivered at the smeltery.

Leaks cannot economically be stopped while the jackets are in operation. As soon as discovered stop the furnace and insert a new jacket. They should be $\frac{9}{16}$ of an inch thick. Jackets after long use absorb silver. The actual quantity absorbed in a year might not be large, yet sufficient to justify smelting the side that comes in contact with the heat. Old matte, slag-pots, and other iron that comes in contact with matte also absorb silver.

Jackets are likely to crack or burn at any point. When near the bottom the cause may be traced to the fact that the cores have not been cleaned out, the water being prevented from cooling off the jacket at that point. Muddy water often deposits silt, forming a crust that causes the jackets to burn at the bottom. To ascertain whether a jacket has anything at the bottom take a common sulphur match, hold the tipped end firmly on the jacket 1 inch from the bottom for a few minutes and see if it ignites.

Frequent tests will enable the operator to determine with reasonable accuracy the extent of the deposit. When this trouble is aggravated close down the furnace and clean out the jackets through the handholes. Some jackets have a tendency to crack around the inner opening of the tuyeres because thin at that place. Others burn about 2 feet above the top of the tuyere; this is due to carelessness. They sometimes burn also near the top from lack of water. Rarely cracking on the outside. Mark the jackets with a raised letter. Designate the middle jackets as "C"; the jacket on the left corner should be marked "A"; the back corner jacket, placed diagonally across, is also called "A". The front jacket on the right and the back diagonally placed jacket should be designated "B". There are two "A" and two "B" jackets in all rectangular water-jacketed furnaces. The "A" and "B" jackets are notched at the bottom to receive a tap-jacket. The length and height vary according to taste. A tap-jacket 12 inches long and 8 inches high is preferable. The hole through which the matte and slag issue should be near the bottom. Beneath it the jacket should be solid. These tap-

jackets are provided with two 1-inch holes, placed in the upper left and right hand corners, for water feed and discharge. Simplicity of construction is to be sought in water-jackets. A perforated pipe will cool the brickwork in front of the tap-jacket and prevent oozing of lead from the crucible, owing to the heat of the slag. In this way a minimum quantity of water is used at exactly the right place. Make the back jackets the same as the front. The tap-jacket hole in the back can easily be bricked up. In starting the furnace this opening is useful for various purposes.

It is impossible to have greater simplicity than with three patterns for jackets. No change in any should be made without authority. The importance of being able to remove the tap-jackets quickly cannot be too strongly emphasized. With each successive enlargement of the furnace the blast pressure is increased. Rapid driving necessarily enlarges the zone of fusion and corrosion of the brick is greater. At one time it burnt out above the jackets, causing the blast to be cut and the tuyeres removed. Finally the area above the main jackets was entirely filled with water-blocks. This

plan worked well. These water-blocks should be 14 inches wide and 30 inches long; each block is provided with a hand-hold. Have a 3-inch feed-pipe with a special valve for each. When properly equipped there is no trouble with this part of the furnace. The water-jacketed mantle (caisson or deck-plate) is one of the most important improvements made in modern lead-smelting. It supports the superstructure, serves the purpose of a water-pipe in supplying the main jackets, takes up less room, and gives the entire furnace a neat appearance. Formerly the mantles were of cast iron, which cracked or burnt out. To obviate the difficulties they finally were built with one channel iron and two "I" beams, riveting over these a sheet of $\frac{3}{8}$-inch steel plate to the top and bottom. This produced a perfect box, with one "I" beam in the centre; this did not extend to the extreme outer ends and allowed the water to circulate on both sides of the centre "I" beam. A box thus built possessed the maximum strength and lightness and could not warp or burn when the water was allowed to circulate. It is imperative to supply the bottom jackets with water. The mantles rest on the corner

supporting columns, with a raised flange to prevent them slipping off and are connected with water-pipes at the corners. They are fed with two 4-inch pipes, one entering the front, the other supplying the sides. Hand-holes are necessary. Have three vents for the air and steam that accumulate at the top of each mantle. They should be tested at 150 pounds pressure before being shipped from the foundry. The small water-blocks on top of the main jackets may be dispensed with by extending them to within a few inches of the mantle plates. This has its objections, but is simpler and cheaper, and materially lessens the number of units and the quantity of water required.

The water-blocks should be run cold, but the bottom jackets as hot as possible. The bottom is always hotter than the top,—a proper condition, particularly at the tuyeres. When the bottom jackets are run cold a train of evils follow, chief of which is the production of cold slag, containing much lead. Anything that produces metallurgical losses should be avoided.

For remote sections, and at places distant from railroads and foundries, jackets of wrought iron,

steel, or various kinds of sheet-metal, are preferable. Otherwise cast-iron jackets are advised, because of the smaller first cost, ease in handling, and facility for quick changes. Jackets of mild steel, hammered steel, and phosphor-bronze are on the market. Solid copper has recently been used with good results.

THE SUPERSTRUCTURE.

All brick used in the lower part of the structure should be the best hard-burned fire-brick. It is customary to use common red brick for the outer walls, but second-grade fire-brick are better. Special pains should be taken to fill every joint full of mortar. The practice of running up the outer walls of a furnace like those of a building, only tipping the outer ends with a little mortar, will prove disastrous. The work should be closely scrutinized as it progresses. At a point one floor above the mantels the furnace walls should have a thickness of 3 feet; the outer walls should be plumb; the inner walls should follow a line running from the top of the main water-jacket to the feed-door; the thickness

of the walls at the feed-door should be 18 inches. Make the entire inner lining of header courses 9-inch fire-brick. These should be tied to the outer walls every 30 inches. Do not have air-channels in these walls, nor should spalls be used for back filling. Furnaces are liable to burn out at any point above the mantels, penetrating the walls at least 2½ feet; sometimes the lower part of the wall may be burnt through.

The corner cast-iron angle-braces should be 1½ inches thick, each side 18 inches wide, and at intervals of every 12 inches there should be placed heavy cast-iron lugs for binding-rods, 1½ inches in diameter. In the centre of each of the side-walls there should be a cast-iron "T," 1½ inches thick, with a wide bearing against the wall, and a centre rib 6 to 8 inches wide. The rods used for binding rest on this rib and form a truss. It would not much increase the cost to encase all the brickwork with cast-iron plates above the mantels; this would add strength and durability. The sill-plate for the feed-door should be 2 inches thick where it covers the top of the wall, but the sides may be 1 inch. In the centre and top of the sill-plate there is cast a lug to hold

in place a bar of 65 pound railroad iron, the flat side out and flush with the outer wall. On this there are short rods with slotted holes to hold in place the movable sheet-iron doors that are securely held in place by wrought-iron wedges. All plates should be securely held with strong bolts. The outside of the ends of the furnace, above the feed-door, should be constructed of specially hard brick. Make provision for the top longitudinal rod, which has gradually increased in size from 1 inch to a full 2-inch rod. A heavy wrought-iron washer is provided for this rod. For the furnace doors there is left an opening on each side 8×12 feet. There is scarcely any brickwork left, except the two upright end walls. The side walls are joined by a cast-iron lintel, 2 inches thick, re-enforced by two pieces of heavy rail.

In case the building be of wood covered with sheet iron, extra precautions should be taken to prevent fires.

The smoke-pipe is 6½ feet in diameter, made of No. 8 steel, closely riveted and provided with a strong butterfly damper, placed 2 feet above the floor, with a strong square stem. This damper is

an important part of the furnace; the handle should be provided with a lock. The pipe rises vertically 2 feet above the cast-iron stack-plate, and then inclines at an angle of $37\frac{1}{2}°$. It then makes a sharp turn downward at a like angle, and at a point 6 feet above the floor it should drop vertically into the top of the brick dust-chamber with a large bootee. Special bands should be provided so the pipe will rest entirely on the iron floor. Close all the pud-lock holes before the scaffolding is removed, clean the brickwork, inside and out. Paint all iron with one heavy coating of graphite paint.

Test the water on all water-jackets; see that all pipes, valves, hose, damper, and lead-well are in good order, and when the last rivet on the pipe has been driven start a fire in the crucible, gradually increasing it for three or four days. The width at the crucible varies 30 to 50 inches; the length has varied from 60 to 160. The water-jackets have had a tendency toward greater bosh, varying from 5 to 12 inches; 12 is excessive, 10 is better. Unless the ore and fuel is coarse, 42 inches wide at the tuyeres is advised. With a furnace 60×160 inches at the tuyeres a powerful unit is created, particularly

if the blast is furnished by four compound condensing blowing engines and four bag-houses. Improvements in construction will doubtless continue each year; still a furnace that will smelt 150 tons of ore each 24 hours on a monthly average would seem nearly large enough. To effect greater economy it must be along the lines of dispensing with manual labor, partially or entirely, for bringing furnace material and taking away the products. The height of furnaces, measured from the centre of the tuyeres to the top of the ore charge, is important. No arbitrary rule can safely be laid down. One 30 feet has been used, but a height of 17 feet is preferred.

Lead ores should not be smelted without a baghouse for filtering the fume; and the matte values that escape through the slag should be collected with an improved slag-settler. Everything, both at the top end and bottom of the furnace, should be saved.

THE POWER PLANT.

From a six months' test it was demonstrated that the cost a horse-power a year was $36.18. This indicates a coal consumption of 3 pounds to the indicated horse-power. This is an excellent showing, considering the fuel used—a low-grade slack.

Managers have had to face numerous vexatious problems relative to the power plant. Not the least has been the relative economy of steam and electricity. A case in point:

Figures were prepared with references to the introduction of a steam plant remote from the smeltery, the engine to be operated by steam from the main boilers, 1100 feet from the proposed engine site. At the same time calculations were made for the installation of a complete electric-power station close to the boilers.

Figures disclosed the fact that the steam plant would cost $4950, the electric-power plant $4815, or a saving of $135 in favor of the latter. The cost was so nearly identical that, for practical purposes, it was immaterial whether the engine was run by

steam at the new plant or steam close to the boilers. The question then resolved itself as to the most desirable location for the future needs of the works, and the relative efficiency of steam and electricity. Comparative figures obtained from qualified steam engineers are appended. The efficiency of electric transmission, based on guarantees, will be 81.5 per cent.; the efficiency of steam transmission, 60 per cent. This indicates a saving of 21 per cent., or, on 200 horse-power, equal to 42 horse-power, by the installation of the electric plant; therefore 42 horse-power, at $36.18, would amount to $1519.56 a year. The wages of an engineer and helper would swell the additional cost by $1700 a year, if steam were used near the plant. Now $1700 plus $1519.56 gives $3219.56, the saving in a year by the electric over the steam plant. It was also calculated that the depreciation of the electric would be materially less than that on the steam plant. This and other considerations favored the selection of electricity as a motive power.

To obtain a broad and comprehensive view of all the different types of engines and their respective

efficiency a variety of data directly bearing on the subject has been collected and adjusted.

A Hamilton-Corliss engine, having a diameter of 20" steam cylinder and 48" stroke, was driving two Baker blowers and one small slag hoist. The engine was running 59 revolutions. The blast pressure was 3 pounds. From indicator card Fig. 10 the horse-

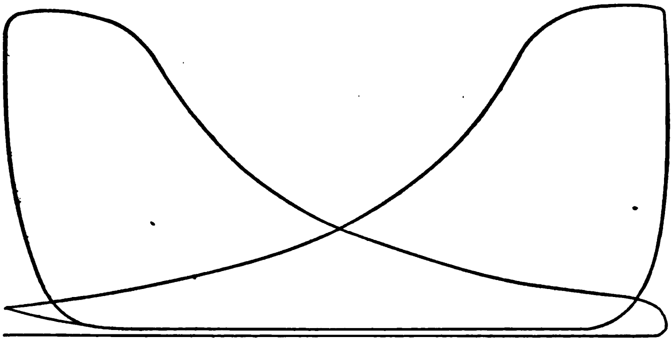


FIG. 10.

power generated was 150. The horse-power of this and all the other engines was accurately determined and the cards read by the use of a planometer.

The same engine was indicated when the blast pressure was $2\frac{1}{2}$ pounds, the horse-power 138. Be-

cause of the proximity of the engine to the boilers, an assumption that the loss in the transmission of steam would be about 5 per cent. was made. On this there would be 157.8 horse-power generated at the boilers. This rated 200 horse-power engine had 50 horse-power in reserve. A consideration of the card itself shows it to be fairly good, but the release on one end could be a little earlier.

The Southwark blowing engine, Fig. 11, which

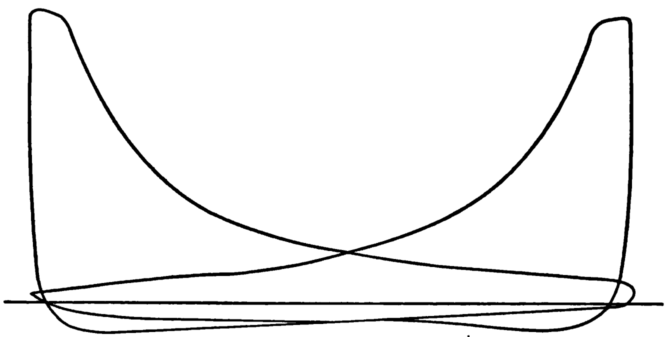


FIG. 11.

has a steam cylinder of 32″ diameter, 60″ stroke, and an air-tub of 84″, was running 50 revolutions at a pressure of 3 pounds. The indicated horse-power was 383. Assuming a loss of 8 per cent. in steam transmission the horse-power at the boilers

would be 417. The indicator card taken from this engine was good, and the vacuum excellent considering the altitude. The vacuum as registered by the gauge was $22\frac{3}{4}$ inches. A large air-pump for condensing is necessary at an altitude of 5000 feet. The card shows the engine was overloaded, and at rare intervals it is necessary to run the engine at a revolution greater than 50, and occasionally up to 60, yet it is not believed it would be wise to run 60 permanently. The question would naturally arise, How can the efficiency of this particular engine be increased? Two ways are apparent. First, increase the resistance of the air passing through the furnaces—which resistance would be greater if they were higher; or, second, increase the size of the air-tub. The engine could be worked in connection with another compounding it, making this the high- and the other the low-pressure engine. This would be more economical. It was made for a high-pressure blowing engine, with a maximum of 12 pounds. It was run without a governor. The repairs were slight, and there was no trouble in running.

The Brown engine, Fig. 12, at the bag-house, with

a steam cylinder of 22 inches diameter, stroke 48 inches, 50 revolutions, driving one 10-foot Sturtevant exhaust fan and one No. 5 Root blower, also

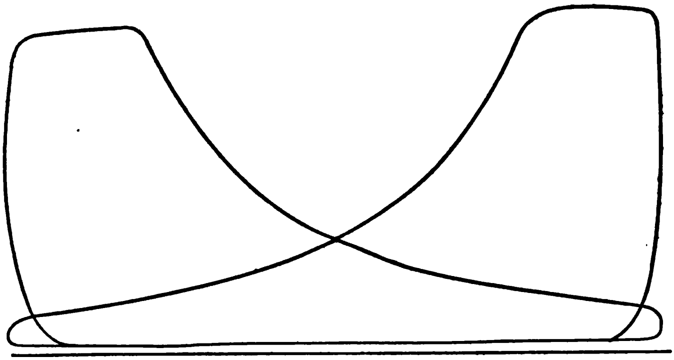


FIG. 12.

one dynamo (Edison, No. 6, current 200 amperes, 115 volts), showed a horse-power of 174.

The various indicator-cards show 1146.6 as the total plant horse-power: that developed by the boilers, 1268; loss by radiation and condensation, 121.4. Considering the distance and the multiplicity of pipes, this is not excessive, and is due to careful steam covering and other appliances for preventing radiation. The horse-power lost by radiation could

possibly be placed at $1300 a year; exclusive of labor and including only the net coal loss.

Attention has often been directed to the importance of a union of all the exhaust steam about industrial plants by the use of a central condenser.

It would be well to make frequent tests of all boilers for efficiency. In the use of both modern water-tube and old-fashion tubular boilers a series of tests demonstrated that the latter, when properly made and set, was 27 per cent. more efficient.

TOOLS AND IMPLEMENTS.

Provide each furnace with a large anvil.

The tapping-bar is an important tool. It is made of ¾-inch octagon mild steel, varying in length from 8 to 14 feet. It formerly was made either with a round or octagon head, but later the head was omitted. When a bar becomes fastened in the tap-hole the head is spanned with a tool having a circular head and an eliptical hole, and into this, above the bar, is inserted a wrought-iron wedge; by striking it the bar generally is quickly removed. The rods used for stopping the flow of lead,

matte, and slag are 5 to 16 feet long. The end is circular, 2½ to 3½ inches, and solid for a foot, welded to a ⅜-inch pipe; at the other end is a circular handle welded to the ⅜-inch pipe. A light, rigid tool is thus secured. A variety of ⅝-inch round iron rods are used at the front of the furnace. Coke rods are used for starting the flow of matte and slag.

Provide every furnace with heavy warehouse brooms, having rattan centres specially made.

Tough, plastic, yellow clay is used for stopping the flow of slag and matte. It should be free from sand or pebbles. It is advisable to keep a supply for use in cold or rainy weather.

Use around the lead-well a short pointed steel bar, ¾-inch octagon, 5 feet long; a perforated skimming-disk, and a wooden hoe-like device for manipulating the copper crust.

The hole at the bottom of the raised lead-well is opened with a pointed bar; the lead comes out hot and smoking; after passing through a short spout it flows into a cast-iron tray, 10 inches deep, 30 inches wide, and about 4 feet long; from this the skimmings are removed with a perforated disc and

dropped into a perforated wheelbarrow. The crust is cut with a garden spade into convenient pieces in the barrow while hot.

When the bullion has been skimmed it is tapped from the tray into 100-pound moulds, the company's brand on the bottom. Twenty bars are cast at a time. The moulds are placed on an iron carriage resting on wheels and running on a track. The bullion usually is cooled with water; the moulds upset and placed on another carriage. It then goes to the bullion-house, is weighed, sampled, numbered, and loaded.

A convenient tool-house is an essential adjunct. In this should be kept an ample supply of tapping-bars, "coke rods," bullion moulds, tuyere cloths, tuyeres, jackets of all kinds, fire-brick, and marl.

At each corner of the furnace provide water-hose for sprinkling and other purposes.

Distances between the furnaces should not be less than 25 to 30 feet. No greater errors have been committed than the crowding of furnaces.

Around all the furnaces 3 × 5 feet iron plates 1½-inches thick should be laid closely together on a 2-inch bed of screened sand.

Each furnace should be provided with a matte-box of 100 to 150 cubic feet, the bottom is cast in a solid piece, like a pan, 8 inches deep. A recess should be cast on the sides of this pan for the side and end pieces. The four sides, forming a box, are securely bound together with longitudinal and transverse rods. No cover is needed, as the slag cover is sufficient. The box rests on two trucks, the wheels having broad flanges. Freedom of motion is secured by using swivels. The railway track should be so laid as to throw the end away from the tapper.

The slag enters the left-hand corner of the box. The slag-spout should be 12 inches wide, and shallow. The tap-hole for the matte is in the side-plate, but sufficiently high to admit a spout, narrowed and semicircled at the matte-hole, but flaring outward therefrom, and having a sharp incline, the outer edge much thicker.

A space of at least 15 feet should be kept between the furnace and the dust chamber for convenience in removing flue-dust and bullion.

A gauge is needed only at the blowing-engine,

except for special runs, say when smelting old slag, or during an antimony or copper run.

The quantity of air entering the furnace is of prime importance, but the pressure required to secure it is secondary. The pressure latterly has been greatly increased by the predominance of ores finer in size.

The main blast-pipe, 5 feet diameter, should be No. 8 sheet steel, closely riveted, firmly hung, and free from angles.

Each furnace has its own air-valve. The handle should give proper headroom. Only the most approved valve should be employed. The bustle-pipe should be 2 feet in diameter.

The upper edges of the 8-inch wrought-iron tuyere pipe should be well rounded to allow the cloth to be easily attached, the cloth 24 inches long, extending over the pipe 4 inches. Trouble with the tuyere cloth increases with the pressure. A heavy, seamless, cotton hose, lined with rubber, meets every requirement. When a standard size has been adopted it should not be changed. The tuyeres should be as short and light as possible, and the points should be cast-iron, but very short; bring the

peep-hole, $1\frac{3}{4}$ inches in diameter, close to the jacket, and use a circular, soft, pine plug, through which has been bored a $\frac{3}{8}$-inch circular hole; close this with cotton waste or paper before inserting the plug. Have the rods mill-turned and saw them into $3\frac{1}{2}$-inch pieces. The end of the plug should be "swedged" down, so it can easily be inserted in the tuyere opening. The tuyere should be made in three parts —cast-iron point, sheet-iron pipe, and wrought end. The tuyere point is turned to fit snugly into the turned hole of the water-jacket. It should be so designed that when slag enters the tuyere there will be a slight outward pitch. The end of the peep-hole should be thickened to resist a sharp rap, even from a sledge hammer. Leakages around the tuyere and the water-jackets are a source of vexation and expense.

The tendency of the main delivery pipe to vibrate may be checked by rigid bracing. The bustle-pipe should be securely attached to the furnace. The vibration is due not alone to the high pressure of the blast, but to the pulsations of the blowing-engines. There are a number of approved blowers that may be used on small and old-fashioned fur-

naces. Blowing-engines should be used for smelting lead ores by blast furnaces at all modern plants.

The feed floor should be covered with ½-inch steel plates with countersunk rivets. Where it joins the furnace, rivet large angle irons to prevent ore or water passing. Have it extend under the sill-plate. Along the front of the furnace, but on the feed floor, run an 18-inch gauge track, elevated 3 feet, for bringing to the furnace the fuel, fluxes, and side ores. Formerly the ore was brought to the furnaces in wheelbarrows, but these have to some extent been superseded by charge-barrows, similar to those at iron blast-furnaces. This is impracticable except where there is a depression in the floor near the furnaces. A better device is a U-shaped ore-cart, with wooden wheels 4 feet in diameter, the spokes and rim of wood, the hub of iron, with a wheel such as is used in a light express wagon, and a half-round tire. The axle, 1¾-inch square, runs entirely through the bed. An ordinary carriage axle will serve the purpose. The vehicle runs and dumps easily on a level floor. With it 1200 pounds of material can be moved by one man close to the furnace door. Limestone cars should

be constructed to permit large pieces to drop from the incline bottom by opening of a hinged door. For handling coke a side-dumping car of 800 pounds capacity may be used. The cars used in handling iron flux and side ores have steadily grown in size. They are of the big scoop type so commonly used in mining.

All plant tracks should be made of 30-pound rails.

The feeding shovels have long handles and square points. There is also a square-pointed, broad-bladed shovel used for throwing coke into the furnace. The use of forks for this or for unloading coke is not advised.

Limestone should be fed in large lumps.

The furnace is barred from time to time with 1¼-inch octagon steel bars; the 10-pound hammers are double-faced, with hickory handles, secured by long, tapering, wrought-iron wedges.

Each furnace should be supplied with a separate 8-beam charging scale. This should be set level with the floor, constructed so the foundation is free from vibration. The scales should be tested every shift and be kept perfectly clean. The platform

can easily be taken up. The room beneath should be ample for the accumulation of ores. The ball-bearings of the scale mentioned give a free swing of the platform.

There should be provided a suitable tool-box for the feed floor; also provide a place in which to keep talley-books and furnace-blanks.

Back of the smokestack there should be a free passage of 7 feet—in this, set the scales, and back of them place two ore beds. These should be at least 100 feet from the nearest railroad track. Construct *three* lines of broad-gauged tracks for the blast-furnace department. Back of the ore beds place the sampling works, and near them the roasters.

BLOWING IN.

Changes are constantly occurring in the methods of starting lead blast-furnaces. The one here described has been subjected to the fullest test.

Kindle a cordwood fire in the crucible and gradually increase the fuel until full. When the crucible is dried out and the plates are so warm that one can scarcely bear the hand on them, and steam

arises from the crucible, there is then melted enough lead to cover the lead-well opening. Generally, in starting an entirely new furnace, it is best to use lead free from silver. Remove all unburnt wood, care being taken not to leave a single spark. Insert the tap-jacket and lute up with moist clay all tuyere openings. Charge from the top of the furnace a cord of wood, throwing it lengthwise. On this place from 4000 to 6000 pounds of coke. In uniform layers insert 20 to 28 charges, consisting of 1200 pounds blast-furnace slag, 150 pounds iron flux, 200 pounds coke. It is unnecessary to kindle a fire in all the tuyere openings. The tuyeres are now firmly inserted and a very small blast turned on. Care should be exercised to run the furnace with a 6- or 8-ounce blast for two or three hours.

Soon after turning on the blast begin with the regular ore charges. Add enough iron flux to make a thin slag; the quantity, of course, will depend on the character of slag used. It is best not to run for the first few shifts on a silicious slag, nor speed the furnace.

When the furnace settles down to good work the

blast is increased and held at 48 ounces pressure, and often with ores fine in size 64 ounces is necessary.

CALCULATION OF CHARGES.

The calculation of charges for blast-furnaces is more a business than a scientific problem. It resolves itself simply to this, How can ores be smelted to produce the greatest revenue?

The laws of chemistry, as well as certain metallurgical conditions, must be respected, but the business features should not be lost sight of. It will be assumed the smeltery is a modern plant and the custom is to bed the ores. These should be prepared with care, to the end that the ores be uniformly spread and that certain necessary conditions be observed:

A. The percentage of silica should be greater than the combined percentage of iron and manganese.

B. Place on the beds as much gold and silver ore as possible, to save interest.

C. Avoid an excess of any one element, such as

calcium, barium, magnesium, aluminum, and particularly zinc.

D. The bed should not be deeper than 5 feet.

E. Lead should not be wasted on the ore mixtures; 8 to 15 per cent. is desired.

Take a sample 36 hours before the bed is completed. After the furnaces have been smelting a certain bed of ore for a short time it is well to take other samples. Bed sampling should take precedence over everything else. The reason is obvious.

The chemist determines the percentages: silica, iron, manganese, lime, baryta, sulphur, and zinc. Longer time is required for the magnesia and alumina determination.

The quantity of roasted material does not widely vary from day to day, but whatever is made should be smelted daily, to minimize the labor required to store it. The quantity of roasted ore varies on the charge, according to the speed of the furnaces. Changes must, therefore, constantly be made in the charge. Use as much bed mixture as possible on the charge. Ascertain before figuring a charge

how much roasted ore and how much bed mixture can be safely used. Years of experience teach that the total charge of ore should be about 1100 pounds, and that as little barren lime and iron flux be used as possible. Use 300 pounds slag on 1100 pounds of ore mixture to keep the dump clean. The fuel (coke) should be as low as possible. This varies from 13 to 14 per cent., according to circumstances. The percentage of coke should be figured on the sum total of the ores and fluxes. On a charge of 1100 pounds the coke approximates 200 pounds.

The silica should vary from 30 to 35 per cent.; the iron and manganese, 23 to 27 per cent.; the alkaline earths, 18 to 22 per cent.; the zinc, 2 to 6 per cent.; the alumina, 4 to 8 per cent.

The composition of lead slag is thus reduced to narrow limits. Taking the mean of the figures as a basis of calculation, there is obtained: 32 per cent. silica; 25 per cent. Fe+Mn; 20 per cent. CaO, BaO, MgO; 4 per cent. Zn; 6 per cent. Al_2O_3.

It is possible to produce a slag with exactly 32 per cent. silica; yet, should the calculation show a

fractional variation, it would not be wise to change the charge.

The iron and manganese, for convenience, is figured in metallic percentage. Chemistry teaches that these elements enter the slag mainly as protoxides.

It has been stated there should be 20 per cent. lime, but it is not meant to add together the percentages CaO, BaO, MgO, but using CaO as a basis, multiply BaO by .35 and the MgO by 1.4.

The zinc in the slag cannot always be controlled, nor need particular attention be paid to its presence until it runs over 6 per cent.; then reduce it by adding some pure ore to the charge. Slags containing 15 per cent. Zn have been made, but are not advised.

The percentage of alumina in the slag cannot always be controlled, but this is not important.

The printed form successfully used for a number of years follows:

Record on the form the figures representing the chemical composition of the materials to be smelted. It is ascertained that 150 pounds of roasted ore,

Charge Calculations on Bed........ Date....................19....

Gross Lbs.	Ore	H_2O	Net Lbs.	Per Ct.	SiO_2	Per Ct.	Fe+Mn	Per Ct.	CaO BaO MgO	Per Ct.	S	Per Ct.	Zn	Per Ct.	Pb	Per Ct.	Al_2O_3
	Ore totals																
	Limestone.. ...																
	Slag...........																
	Coke ash..																
	Scrap iron......																
	Total ore and flux...........																
	Deductions.................																
	Grand totals																
	Factor for calculated slag...																
	Slag actually found.........																

RÉSUMÉ.

Coke per charge.......................
Scrap iron... ....
Net per cent. lead ... .
Grade of bullion ... .
Average assay of slags.................
Ore tons per furnace...................

SiO_2..................................
$FeO+MnO$..... ....
CaO....................................
ZnO. ....
Al_2O_3.................. ....
Other ingredients... ....
Factor....... .:.... ....

MEMORANDA:

Charge Calculations on Bed. **First Calculation.** **Date............19....**

Gross Lbs.	Ore	H_2O	Net Lbs.	Per Ct.	SiO_2	Per Ct.	Fe + Mn	Per Ct.	CaO, BaO, MgO	Per Ct.	S	Per Ct.	Zn	Per Ct.	Pb	Per Ct.	Al_2O_3
650	Bed 7399......	4	624	20	124.8	7	43.7	15	93.6	6	37.4	3	18.7	18	112.3	9	56.2
150	Cint. ore......	0	150	11	16.5	30	45.0	0	00.0	4	6.0	6	9.0	19	28.5	0	00.0
100	Aspen D. P...	3½	97	8	7.7	2	1.9	40	38.8	0	0.0	0	0....	0	0..	0	0
200	Iron flux.......	15	170	10	17.0	50	85.0	0	....	0	.... ..	0	....	0	.... ..	0	
1100	Ore totals............		1041	....	166.0	....	175.6	....	132.4	....	43.4	....	27.7	....	140.8	....	56.2
0	Limestone.................																
300	Slag.......................			31.5	94.5	26	78.0	20	60.0	1	3.0	2.8	8.4	....	... ..	5.8	17.4
	Coke ash..................			....	18.0												
	Scrap iron........																
	Total (ore and flux)........				278.5		253.6		192.4		46.4		36.1		140.8		73.6
	Deductions.............................				..		30.8		....		⅝=30.8		80%=		....		
	Grand totals..........................				278.5		222.8		192.4		....		28.9		....		73.6
880	Factor for calculated slag...........				31.6		25.3		20.8		....		3.2		... .		8.3
	Slag actually found......................																

Coke per charge..................	200
Scrap iron.........................	0
Net per cent. lead..................	
Grade of bullion....................	
Ore assay of slags..................	
Ore tons per furnace...............	

SiO_2..............................	278.5
FeO.................................	286.4
CaO.................................	192.4
ZnO.................................	36.1
Al_2O_3............................	73.6
	867.0
Other ingredients...............	13.0
Factor....	880.0

MEMORANDA:

for example, can safely be used; record the analysis of this in the lines prepared for it.

The quantity of bed ore used to a 1100 pound charge varies from 450 to 650 pounds. For commercial reasons try, first, to use the maximum quantity—650 pounds. Multiply the net weight, 624, by the percentages of composition of this bed. Place the totals in the spaces provided.

Multiply 150 by the percentages found in the roasted ore. The charge can be made more profitable by dispensing with limestone and using a lime ore. Ascertain how much of this can be used from the stock on hand, then complete the calculation.

Merely as a trial figure, 200 pounds iron flux is set down; multiply the net weight by the given percentages, and add the pounds of SiO_2, Fe, Mn, CaO, BaO, MgO, sulphur, zinc, lead, and alumina.

The last analysis of the slag is taken. For the 100 pounds of ore mixture there is used 200 pounds of coke, containing ash, 17 per cent., or 34 pounds in the 200; but 60 per cent. of this ash is silica, as found by analysis; hence, $34 \times .60 = 20.4$ pounds silica. But there is a small quantity of iron and alkalies in this ash, and 18 is the calculated

quantity of silica not fluxed. Then again add the columns.

Practically all the silica will pass into the slag. All of the iron will not, since a part will unite with sulphur, forming matte. Some of the iron passes into the slag, forming calcium sulphide. Some remains in the furnace, forming accretions. Some forms fume. Two-thirds of the sulphur enters the matte. Deduct two-thirds of the sulphur from the iron.

Practically all of the alkaline earths enter the slag as such. Of the zinc, 20 per cent. goes into the fume, some into the hangings, some into the matte, and the balance, or 80 per cent., enters the slag.

Many years of recorded experience developed these facts. It is not a thing that can be figured, but a fact demonstrated by observation and based on careful weights. Substantially all the alumina enters the slag as such.

A résumé of the slag-forming constituents shows: SiO_2, 278.5; $FeO+MnO$, 286.4; ZnO, 36.1; $CaO+BaO+MgO$, 192.4; Al_2O_3, 73.6; total, 867 pounds. There are also small quantities of other slag-forming constituents, including copper, phosphorus, potash,

etc. These amount to about 13 pounds on a charge of 1100. This is ascertained in several ways. The safest rule is to observe the difference between the slag figured on and that actually produced. Add this 13 to 867, obtaining 880, which is the factor for calculating the slag.

Divide the pounds of silica, iron, lime, zinc, and alumina by 880, and multiply the products by 100, giving: 31.6 per cent. SiO_2; 25.3 per cent. Fe+Mn; 20.8 per cent. CaO+BaO+MgO; 3.2 per cent. Zn; 8.3 per cent. Al_2O_3. The figures indicate this to be a good charge. However, it falls far short of being such. It contains too much fine ore (particles small) for the use of so much bed, meaning loss of tonnage. The bed contains 6 per cent. sulphur, which would make the matte production too large. Again, consider that the bed contains 18 per cent. lead—an expensive luxury. The bullion would be too low grade for so much lead. The excess of alumina would cause the furnaces to smelt slow. Experience teaches that 8.3 per cent. alumina is too high for speed.

The quantity of slag-forming material in the charge is somewhat below the average. The rem-

edy is to cut the bed, raise the lime dump (since this is coarse ore), and add 40 to 80 pounds pure silicious ore. Put this altered charge on the furnace, as it meets commercial and metallurgical requirements, and when it has run a few hours take a laboratory sample and make an analysis for SiO_2, Fe, Mn, BaO, CaO, MgO, Zn, and Al_2O_3. The results usually agree closely with the slag produced.

If the iron flux is expensive, lessen it after a few hours' run. A determination of the silica, iron, and lime will enable the operator to balance the charge, but it will be only after years of close observation that the metallurgist can approximate the composition of slag by its appearance when hot. Rarely is a charge made that may not with profit be altered. Changes usually are in the direction of improvement, viewed either from a commercial or a scientific standpoint. The charge on a new smelting mixture should have the silica high and the lime low. The safe side should always be taken. The cintered ore and other constituents are apt suddenly to change in composition. Unless entirely satisfied that the data is correct have additional

analyses made. Analyze the cintered ore several times a week. The office books are presumed to give accurate information regarding silica and iron dumps, but their disclosures should not blindly be accepted, owing to the irregularities of the dumps. From the face of the dumps and ore beds take check samples.

Once a week test the limestone for silica, iron, and alumina. Also as frequently take a sample of the coke and make an ash determination, occasionally analyzing the ash for silica, iron, and alumina, especially if different grades of coke be used.

To summarize: make a preliminary calculation as to what is likely to produce the best financial results; then alter the test charge to conform with known chemical and metallurgical facts.

DAILY REPORTS.

A complete record should be made of every furnace every twelve hours. The day shift should be from 6 A.M. to 6 P.M.; the night shift from 6 P.M. to 6 A.M. Drafts of the forms to be used should be made by the superintendent and metallurgist, care being taken to provide for the preservation of every

physical, mechanical, chemical, and metallurgical fact developed in the operations. The reports may be classed thus: blast-furnace reports, roaster reports, refinery reports, parting-plant reports. A variety of other reports are daily used, but these need not be enumerated. Enter the regular blast-furnace charge in a small blank-book, preserving uniformity throughout the records. Appended is a sample entry:

600	pounds	bed ore,
250	pounds	cintered ore,
30	pounds	silicious ore,
220	pounds	iron flux.
1100		

120 pounds limestone,
300 pounds slag,
200 pounds coke.

Immediately below the charge state whether any furnace is to be blown in or out, and whether there be any side ores, together with all other necessary instructions.

Personally hand the charge to the foreman, read it over to him carefully, and have him read it aloud to

the metallurgist, that he may comprehend precisely what is required. All the transactions should be embodied in the daily reports.

When a new department is added to a plant one of the first things to be considered is the form of reports. Explicit directions for printing and ruling each form need not here be given. Experience is the best teacher.

Reports should not only be carefully preserved, but frequently consulted and compared with results of subsequent operations. A repetition of former mistakes may thus oftentimes be avoided.

A prime essential is a supply order-book, in original and duplicate, with corresponding numbers, and perforated. The stub, approved by some one in authority, constitutes the purchasing agent's voucher.

The thrift of an industrial plant can safely be judged by a careful examination of the daily report blanks.

SUPERVISION AND OPERATION.

Remarks that follow apply generally, and in some instances specifically, to the handling of the lead blast-furnace after it has been successfully blown in.

Run the jackets at all times hot, watching them with much care the first few days. Punching the tuyeres too often leads to long noses. They should be opened only once a shift, to the end that the furnaceman who comes on duty gets an idea of the conditions that have obtained. It is proper that the men on duty at change of shifts should advise their relief how the furnace has run, what jackets are cold, and when the matte-box was changed, with information as to the slag—whether hot, brittle, curdy, tough, or stringy—and whether the slag tapped easily. Quick and reasonably accurate ideas about troubles can be formed, but before changing the charge it is always well to observe the shells of the slag which have recently been dumped. From the liquid and the partially solidified slag the operator will be able to make a quick change in the charge, saving time, trouble, and much money. As

a check, however, an analysis should be made as quickly as possible.

The coke car should hold four charges of 200 pounds each. This is dumped in a bin from an elevated track on each side of the furnace; close to the coke bin the lime bins are placed, there being one on each side of the furnace. The limestone is dropped from a car running on the same track used for coke. This may also be used to bring to the furnaces iron flux, scrap-iron, and side ores from the various dumps. The ore charge usually is about 1100 pounds; the limestone will vary from nothing up to 250 pounds according to the purity of the ores, and whether they contain alkaline earths. Hold the quantity of slag at some fixed point, giving preference to 300 pounds. The coke also should be nearly a fixed quantity, using about 200 pounds. This will necessarily vary at times, but it is better practice to vary the amount of ore than to change the coke. Frequently the charge will be nicely balanced and the slag have the right chemical composition; yet it may be a trifle too cold, or fine ore may sift down, giving rise to trouble when tapping. In the spout lumpy ore

may sometimes be observed, and the matte-box filling with fine ore. In such cases the trouble may be corrected by feeding extra coke, say three cars, extra to the shift. Where there is a persistent tendency toward shifting, it is well to state on the charge-book that every twentieth charge shall be a "slag charge." The extra slag and coke act as a corrective. Sometimes, however, fine ore will run out of the tuyeres, and to correct this there is nothing so effective as additional fuel and slag. Fine ore will rarely make itself manifest at the tuyeres, except when the furnace is hanging badly, when insufficient fuel is used, and the furnace is driving fast. When this condition obtains the best results are secured by using a cutting slag, with plenty of fuel, and then after one or two shifts give the furnace a good top-barring.

By the term cutting slag is meant one that will remove the obstructions. Usually it is a hot, pure slag, low in lime and zinc, but rather high in silica and iron. It is difficult to make a general statement about a cutting slag that will apply in all cases; yet, the success or failure may be said to depend on their proper use. Knowledge of the

chemistry of the furnace is essential to a proper understanding of how and when to use them.

Assume, as an illustration, that the furnace is driving slowly; that the blast comes through the charge irregularly; that it settles unevenly, and that the whole top has a tendency to agglomerate; the jackets take but little water, and the slag is smoky and unsightly. Under such circumstances there is probably an accumulation of sulphides. These are often largely mixed with zinc sulphide, and it is likely the easy fusibility of the charge is caused by fritting of many sulphides. The lead in the roasted ore may also materially cause a low fusing-point. Under such conditions the remedy is in roasting the ore better, in lessening the quantity of lead in the roasted ore, and increasing the quantity of raw matte on the charge. It takes at least thirty-six hours for the ore to pass through the roasters, and there is always a portion on hand that has to be smelted before relief can come through this source. This will require abou' fifty hours; but the furnace needs immediate relief, and unless something is done it may stop. The remedy is: put on a special charge; run the silica at 31 per

cent., the iron and manganese at 28 per cent.; the lime should be materially lessened, but it is impossible to say to what point, owing to the zinc. Do not make the mistake of having the lime too high. Lessen the amount of sulphur as fast as possible; also decrease the zinc sharply by the use of pure silicious ore. It is generally best to lessen the lead. Until the new charge comes down, hold the ore column quite low and never smother the furnace, allowing the blast to pass through freely. Where this matting tendency is noticed use large pieces of scrap iron, in order to produce a quick chemical heat within the water-jackets. There should not be too much economy in the use of fuel; it should not be high, as the chemical result desired is to decompose the sulphides quickly with iron. If this diagnosis be correct a magical change will be noticed. The furnace will begin taking the blast more freely, the charge will in a few hours settle more uniformly, and the slag becomes boiling hot. Much sparkling matte will be formed; the furnace breast becomes soft and the slag can scarcely be held. At first it smokes badly, but as the accumulated sulphides are washed out it clears up. The jackets become

heated, many or all of the lost tuyeres are regained, the tonnage of the furnace is restored, and by the time the new roasted ore comes to it it is freed from obstructions and is running fast.

The analysis of the slag should serve as the main guide, but the operator who uses it *only* cannot be successful, as mechanical and chemical changes are constantly taking place in the furnace not indicated by laboratory data. Special charges, unless absolutely essential, are not advised. Frequent changes in the charge are not advisable. When the furnaces are running well and fast, and the slag is good (low in lead and silver) the aim should be to maintain the charge. The most valuable asset of the metallurgist is specific knowledge of the chemical properties of the various slags, to the end that he may *profitably treat every known mineral* and supply every demand of the industrial world.

A slag high in iron runs fast, produces a large tonnage, and generally is clean in both silver and lead, provided the lime percentage is not too low. A fast-running slag is not necessarily profitable. In it is lost valuable iron that should be used to flux silicious ores. *Tonnage should not be increased by*

the use of expensive iron flux. The free use of iron causes the matte to run low in lead; produces much speiss when there is much arsenic in the ores, but this speiss is partially obviated by using a high silicious slag. Objectionable substances are not annihilated by sending them into the bullion, the matte, and the fume. Arsenic sent into the bullion increases the cost of working it. That sent into the matte, however, is a benefit, since there are fewer crushers broken, the rolls do not wear out so rapidly, and the speiss, when thus mixed with the matte, is in excellent form to allow the arsenic to be expelled in smoke. Speiss piles often contain much gold and silver. The best practice consists in closely working up by-products. Do not have an accumulation of slag, matte, flue-dust, speiss, furnace-bottoms, furnace-cleanings, copper bottoms, fire-brick containing values, or anything else that contains lead, gold, silver, or copper.

INFLUENCE OF METALLIC ELEMENTS.

MANGANESE.

EXCESSIVE amounts of manganese are uncommon. The slag produced is clean in both lead and silver and it creates no furnace difficulties. Twenty years of close observation and careful investigation failed to disclose any relation between the percentage of manganese in slag and its lead and silver contents. Manganese ores are particularly advantageous in the treatment of zinc ores, by reason of the easy fusibility and fluidity imparted. A number of experiments proved that manganese lessens the production of matte to a marked degree. Numerous tests made with matte produced at a number of smelteries failed to disclose the presence of manganese in appreciable quantity; and the little found could doubtless with safety be attributed to the mechanical admixture of slag.

These facts seemed to justify the author's state-

ment, made in 1899, that "It is not known whether it is possible to have manganese in mattes existing as a sulphide. I very much doubt it." A year later public mention was made of a matte produced at Nelson, B. C., said to contain 6.08 per cent. manganese.

ALKALINE EARTHS.

The ore-bed mixtures should always be carefully tested for the alkaline earths, lime, baryta, and magnesia. Neglect to do this often results in trouble and irregularity, particularly when they are not known. There is no element present in such quantities as seriously to interfere with lead-smelting, and *there is no metallurgical operation known which is susceptible of such wide ranges.* It is, however, the most difficult of all metallurgical operations, and requires a more intimate knowledge of the chemical elements than any other department of metallurgy.

Sometimes the ores to be smelted contain so much lime it is impossible to use any limestone. When it is necessary to smelt excessively large quantities of alkaline earths, add silica and iron, thereby re-

ducing the percentage of these bases. Iron cannot be added as a diluent without silica, as calcium oxide and protoxide of iron do not unite. Alkaline earth slags give rise to much trouble where the silica is low and the lime and iron bases are high, and with a sudden drop in the silica they become infusible; they are fluid up to a certain point then become curdy. The slag breaks short and cannot be drawn to a string; the drops are round, fall quickly, and form a heavy shell; they quickly chill the matte-boxes; are bad for the tuyeres; generally tap hard, and, if something is not done, produce the phenomena called "freezing up the furnace."

When silicious ore is very scarce the use of considerable slag will relieve the temporary embarrassment. It will be found expedient to temporarily change the roaster charge, decreasing the quantity of iron, and roast as much silicious sulphides as possible.

A certain quantity of lime in the ore is advantageous since it increases profitable tonnage.

BARYTA.

Baryta ores have been handled with signal success by using a slag running 35 to 37 per cent. silica and 27 per cent. alkaline earths. A lengthy series of tests on baryta ores resulted in a slag containing 14 per cent. barium oxide. The slag was highly silicious. There was only a small quantity of diffused barium sulphate; practically all of it was in combination with silica. Barium sulphate is first reduced to a sulphide chiefly by incandescent carbon. This sulphide is decomposed by silica at a high temperature, forming barium silicate, which passes into the slag, and a portion of its sulphur unites with iron, lead, and copper, forming a matte. Under certain favorable conditions a small part of the sulphur, originally united with barium, will pass off as sulphurous oxide in the furnace gases, but by far the greater part of the sulphur, possibly 90 per cent. of it, is to be regarded as matte-forming. The green flame is due to barium, and when it is quite high the pyrotechnic display, on tapping the slag, constitutes a beautiful sight, particularly at night. A slag low in silica and

high in iron is not best for the treatment of ores containing much baryta. Baryta is not as strong a base as lime or magnesia. These slags are specially heavy, and high silicious slags are necessary to lessen their specific gravity, enabling the matte to settle well. Where much baryta ores are handled roast the sulphide ores down to 2 to 3 per cent., but, in the absence of baryta, 4 to 5 per cent. is low enough. A large roaster tonnage is recommended when the barium sulphate is low. Under no circumstances roast below 2 per cent. sulphur, owing to losses of gold and silver, but to turn out a product as high as 5 per cent. sulphur from the roasters when the baryta is high is faulty, owing to the great quantity of matte produced.

SULPHUR AND MATTE.

For many reasons a production of 10 per cent. matte is advised. Ores can be treated more rapidly by blast-furnaces than by roasting-furnaces, and when is considered the losses that take place in lead, gold, and silver by roasting, treating sulphides by the blast-furnace is cheaper. It is unwise to

roast any ore unless the total sulphur approximates 12 per cent. There are many ores containing more than 12 per cent. sulphur that should be put on the ore smelting beds on account of the large gold and silver contents. Generally speaking it is not wise to roast any ores, however high the sulphur, where the precious metals are large, because of the fume losses and also because of the interest account. A close watch should be kept to prevent the sulphur from varying widely from 3 to 4 per cent. on smelting beds. Where the production of matte is low the silver in the slag is apt to be somewhat high, but it does not follow that the silver in the slag is in direct proportion to the quantity of matte formed. It has some bearing on the subject, but the *kind* of matte formed is a matter of the utmost importance, and the presence of copper in the matte always tends to lower the silver in the slag. When the silver is running high in the slag increase the copper in the matte. This never fails to lessen the loss of silver. With much sulphur on the charge the fuel can be reduced to 10 or 12 per cent. with satisfactory results. This saving is of great importance. Coke is the chief item of cost in smelting. When making

10 per cent. matte the tonnage of the furnace is much greater, the slag keeps hotter and the tuyeres brighter, but the furnace will form wall accretions quicker. Slag-tapping is better with considerable matte; the slag-spouts and matte-boxes are kept in better condition. With a yield of 10 per cent. matte the Iles slag-settlers do satisfactory work. It is well to introduce regularly some matte into these furnaces, aside from the matte prills and globules separated from the slag, in order to keep the bottoms in good condition. This is particularly advisable where the slag is high in zinc, baryta, and magnesia. In advising the production of 10 per cent. matte the general welfare and smooth working of the entire plant has been considered. Formerly it was deemed best to make 4 to 6 per cent. matte. Before the development of gigantic matte-boxes and large slag-settlers the facilities for handling matte were crude and insufficient in capacity. For a long time slag and matte were handled in pots of 250 to 300 pounds capacity, but subsequently they were increased to 3000 to 7000 pounds, operated either by steam or electricity. There were often

long trains of these, representing many tons of molten slag.

By making 10 per cent. matte extremely impure and refractory ores can be treated with greater success. While 10 per cent. matte is desirable in most cases it should not arbitrarily be followed. From 4 to 7 per cent. is sometimes sufficient, especially when treating pure silicious ores and where the barium sulphate is nearly absent. There are times when it is impossible to keep the yield of matte below 12 to 15 per cent. for short periods.

MAGNETIC SLAGS.

It has been found that many lead slags, even though free from matte globules, are *not to any degree magnetic, particularly if powdered in an agate or Wedgwood mortar.* The rubbing off of the iron of the bucking plate and muller imparts the supposed magnetability to these slags. In copper metallurgy, where the roasted ore is melted down, producing a first matte and a slag running 40 per cent. silica, it has been shown to be magnetic and it is assumed that a portion of the iron exists in the form of magnetic oxide.

MAGNESIA.

Once it was thought magnesia was a source of much trouble. Great stress was laid on the high fusing-point of the silicate. This temperature called for an increase of fuel and silver losses were supposed to be high; the mixture of zinc with magnesia was believed to be specially injurious. Experience taught that magnesia gives little trouble when intelligently handled. Magnesia is a strong alkaline base, stronger than lime and four times stronger than baryta; it unites readily with silica, and when this silicate alone is formed the temperature employed is rather high, but when in the presence of a number of other bases it does not materially increase the fusing-point of a slag. *As it saturates much silica the quantity of limestone required is wonderfully lessened. It does not cause silver to enter the slags.*

Magnesia slags are generally low in lead. It forms no hangings or crusts; does not injure the jackets or the tuyeres, nor increase or decrease the matte formed, and when an excess of limestone is not used no difficulty will be encountered. Magnesia should always be figured 1.4 times stronger

than lime. It produces a hard slag resembling stone, and makes the crystals small and few in number. Much magnesia, zinc, and alumina are unpleasant combinations, but can be mastered by regulating the lime that enters the slag. Lead entering the slag should influence the quantity of alkaline earth used. When the slag runs very low in lead look for stormy times, as it is apt daily to increase in lime and suddenly give rise to trouble.

ALUMINA.

The presence of excessive quantities of alumina need occasion no alarm. Aluminous slags generally run slow; high blast will help along matters. It is best to keep the iron well advanced, and the silica 29 to 31 per cent., with high alumina. Alumina is exceedingly adaptable. There are times when it seems to favor the entrance of lead in slag; at other times the charge may be low in lime, yet the slag will be low in lead. These conditions may continue three to five days, when suddenly the lead will increase. A sharp addition of lime, iron, and fuel is then recommended. Alumina has no effect on the silver in slags.

WALL ACCRETIONS.

Blast-furnaces used for smelting lead ores do not, as a rule, smelt rapidly when first started, and the tonnage increases at a certain time, as wall accretions are formed. The furnaces do not begin to smelt ores rapidly for two or three weeks. There are times when the tonnage shows a material increase; then less fuel is needed. When wall accretions grow too large irregularities appear. It is a matter of importance to know when to remove them. No arbitrary rule can be laid down, since it is dependent on the quantity of zinc, sulphides, and sulphates of other metals in the ore. Considerable importance attaches to the general handling and regulation of hangings. When regularly formed on the walls, and not too thick, the effect on the furnaces is good; otherwise bad results follow: the furnaces run slow, producing much flue dust and fume. These hangings occasion considerable losses of silver, lead, and gold, unless the plant is provided with a bag-

house. When the hangings are heavy there is often obtained an action like the oxidizing flame of an enormous blow-pipe; at these times there is noticed a roaring noise, much crackling of limestone and sulphide, dust is raised, and the smoke is white, especially if the charge be allowed to drop but a few feet. The tonnage is sometimes great, yet, if an accurate account of all the furnaces is kept separately, it will be found the *grade of the bullion in silver is greater on a "hanging" furnace, the quantity of matte is smaller and higher in silver.* There are two distinct ways to remove hangings. "Short barring" usually is practiced once a month; sometimes at shorter intervals, according to the quantity of zinc and lead sulphides treated. Feeding of ore ceases, and, as the charge descends, the blast is lessened. When the charge has reached the top of the water-jackets the tuyeres are removed, 2400 pounds of coke are added, and, when the smoke has partially cleared away and the tuyeres luted up with thick clay, the barring of the wall accretions should begin. For this purpose various lengths of 1½-inch octagonal steel bars are used. Dislodge loose top material quickly and cut off

overhanging pieces; remove the inner shell first, the accretions back of it will be found less compact. Time may be wasted in trying to take off too large pieces. As the barring proceeds scatter coke through; when in condensed form scrap-iron is serviceable, but in the absence of this sprinkle 2 tons of iron flux through the barrings. From 8 to 10 hours is long enough to suspend smelting for barring. Do not remove all the hangings, even if possible, but straighten them; take off the large knobs and leave the wall smooth. Turn on the blast before beginning to feed the furnace, as there is often difficulty in forcing the blast through it on account of cohesions. When the blast refuses to come through continue to increase it, and drive long bars into the hangings. If coke is sprinkled through the hangings during the operation this trouble will not arise. Frequently it will be found necessary to repeat this short barring after the furnace has run 12 hours, particularly when the top hangings fill the furnace before the lower hangings can be reached. Care should be exercised not to leave a large projecting shelf, as that would cause the furnace to hang again quickly. Short barrings are

not always effective. When the hangings become too heavy the best way is to blow out the furnace; "feed it down" with finely crushed limestone. Continue this, in order to lessen the flame, until the slag ceases to flow; then shut off the blast, and immediately take out the two front jackets, breaking down the crust with heavy bars. Pull out the hot coke, slag, and fine limestone with a long roaster hoe, and wet down the furnace materials. Sulphuretted hydrogen and arseniuretted hydrogen (the latter an extremely poisonous gas) are produced, and care should be taken not to inhale them. As quickly as possible take down the two back jackets, brace the sides to prevent spreading, and cut through the crust. When loose material within the jackets is well cleaned out begin barring from the top, use plenty of water and force down the dust and other loose material. After the loose hangings have been removed suspended platforms become useful. Have two of them made of 3-inch plank, bolted together with heavy chains, and securely braced to the walls of the furnace. The hangings may then be chipped off with wedges and gads. While this is proceeding the lead in the crucible solidifies. After the

hangings have been removed and the hard crust below the tuyeres cut out, dig a 16-inch hole through the crust, reaching completely to the solidified lead. From the bottom of the water-jackets to the feeding-door the furnace is now clean. The solidified lead in the crucible is covered with a hard crust, penetrated only by the hole. Now put up the front and back jackets; add a few sticks of cordwood and fill the furnace with coke to the top of them; from this point fill the furnace with slag charges, using 1200 pounds slag and 150 pounds coke, interspersed with an extra 2400 pounds of coke. When the furnace is filled to about 7 feet of the door throw in 6 to 8 regular ore charges, put in the tuyeres, and light the coke through the peep-holes. Giant powder and mine drills have successfully been used for the removal of furnace crucibles, but they cannot be employed to remove wall accretions.

HANDLING OF SMOKE.

THE smoke that arises on tapping slag and matte is injurious to health; hence there should be provision for its removal. Elsewhere is given the chemical composition of it, showing it to be worth saving. A large ventilator over the tap-hole of each furnace, or one large and continuous ventilator over a series of furnaces, however good the construction, is not sufficient. The most effective method is to have the front of the blast furnaces boxed in with sheet-iron. The box should be the full width of the front, at least 10 feet high. Attach it to a 3-foot circular pipe and connect with a 5-foot horizontal pipe. Over all slag-pots, matte-boxes, or matte-trays there should be hoods a trifle larger than the vessel from which the smoke is to be removed. Provide in the up-take pipe a weighted damper, easily and quickly operated. Even where the pipes in front are large they do not prove entirely successful, especially where depend-

ence for draft is a stack, unless it be hot. The pipes over slag-pots and matte-vessels should be made telescopic, and the smaller or inner pipe should fit loosely into the larger pipe and have ample play.

There should be two large, specially constructed fans to remove the smoke from the blast-furnaces.

They should be sufficiently large for present and prospective needs.

For a blast-furnace having a cross-section at the tuyeres of 42 inches × 140 inches, the pipe that removes the smoke should be at least 6½ feet diameter. When smaller a part of the smoke is lost. The importance of saving every portion of it cannot be overestimated. Condemn complicated and expensive doors. Those held securely by wedges of iron or wood are best. Even the large doors on the side of blast-furnaces should thus be held.

Butterfly dampers are rarely a success, but in many places it is difficult to dispense with them. Where used, make a detail drawing of their construction. These dampers should close at a slight angle; the stems and handles generally are too small.

Pipes entering brick walls or flues should have a large bootee, as well as where they join at right angles.

Keep the sheet-iron well painted and avoid all leakage. Where the flues are brick avoid pilasters on the inner walls; make the walls as smooth as possible and the turns gradual; pilaster outer walls. Bind the small and hot flues with a rigid railroad iron, the rods on the outside. Do not for economy make a flue too small. Bind the top arch of large exposed flues with two horizontal lines of railroad iron, 30 to 60 pounds to the foot, place between the rails a rod $\frac{7}{8}$ inches to $1\frac{1}{2}$ inches diameter; encase this rod in a pipe. Place rails on both sides and at the skew-back. Keep the tops of the dust-chambers covered, so they will not disintegrate from snow and rain. Flues should be constructed so they can be cleaned at all times without shutting down. Even in brick flues this can easily be done by having a track and car run underneath. Double flues are desirable. Circular steel pipes of large diameter are generally better than brick dust-chambers, particularly where fans are used.

It was determined that the average analysis of

the smoke from blast-furnaces smelting silver-lead ores was as follows :

Cubic Feet in Each 100 Feet of Fume.		Weight in Each 100 Feet of Fume, in Lbs. Troy, at 34 Degrees Temperature.
CO_2	18.14 cu. ft.	2.089891 lbs.
CO	7.64 "	.554095 "
O	3.41 "	.263024 "
N	68.99 "	5.048688 "
Weight of dust per	100 "	.038000 "
Total weight		7.998698 lbs.

There was practically a weight of .08 of a pound for each cubic foot. The weight of the atmosphere under similar conditions is .0753 pounds a cubic foot; hence the fume is only 6 per cent. heavier than the atmosphere at the same temperature and pressure, and offers 6 per cent. more friction than the atmosphere in similar ducts. Because this smoke contains particles of lead in the fume state, and existing chiefly as sulphide of lead, one would naturally think it would be more than 6 per cent. heavier than the atmosphere, yet it can be stated that such is not the fact.

The formula for loss of pressure in conduits is $P = \frac{Kv^2S}{a}$, in which P equals the loss of pressure in

pounds to the square foot; *V* equals the velocity in feet to the minute; *S* equals area of wall surface in square feet; *K* equals the coefficient of friction; and *a* equals the area of cross-section of duct in square feet. The value of *K* for brick ducts, as determined by Mr. Daniel Murgue, the result of twelve experiments, is .0,000,000,019. For the purpose of calculating the loss of pressure caused by friction of the fume through conduit, add 6 per cent. of this coefficient and it becomes .000,000,002.

A 10-foot fan was observed to be running under these conditions: Speed, 300 revolutions. This fan was removing the smoke from 5 blast-furnaces, each of similar dimensions, and forcing the fume through muslin bags; each of these was 18 inches when distended and 30 feet long, closed at the upper end. Smoke entering the lower end was forced by the fan through the cloth, which strained out, with almost mathematical precision, the solid particles from 5 furnaces. The solid matter was sulphide, sulphate, and oxide of lead, oxide of zinc, arsenious oxide, and a small quantity of carbon. The volume of fume passing the fan at a certain time was estimated to be 186,000 cu. ft. per minute. The bag-house engine

was making 52 revolutions, the fan 305. The horse-power of the engine, taken a few days previously under similar conditions, was 140; actual horse-power in the moving fume 38. Allowing 30 for driving a No. 5 Root blower and turning the shafting, the efficiency of the fan was 34.5 per cent., indicating a loss of 65.5 per cent. The efficiency of the fan was taken when the volume of fume passing was 126,900 cu. ft. Vacuum pressure, .51 ounces; positive pressure, .41 ounces; total, .92 ounces to the inch or 8.28 pounds to the foot; work done by fan, 1,050,632 foot-pounds or 32 horse-power; indicated horse-power of engine, 99; efficiencey of fan, 32 per cent. Twenty-four compartments were in use, representing a filtering area of about 291,600 square feet. This shows an average of .64 of a cubic foot is passing through each square foot of cloth each minute.

METALLURGICAL RESULTS.

As to chemical determinations on slag, it was discovered by taking the complete analysis that the sum total of the SiO_2,$FeO + MnO$, $CaO + BaO + MgO$, ZnO, and Al_2O_3, was nearly 100 per cent. Forty determinations on aluminous slags showed: SiO_2, 31.35 per cent.; $FeO + MnO$, 35.07 per cent.; alkaline earths, 20.25 per cent.; ZnO, 6.23 per cent.; alumina, 7.07 per cent.; total, 99.97 per cent. Rapid commercial analyses have about the following errors: SiO_2 is five-tenths per cent. high, Mn three-tenths per cent. high, BaO approximately correct, ZnO four-tenths per cent. high, Fe five-tenths per cent. high, CaO seven-tenths per cent. high, MgO two-tenths per cent. high, Al_2O_3 four-tenths per cent. high. Losses of gold, silver, and lead occur at the top and bottom of lead blast-furnaces. Experiments covering a period of twenty-five years prove that these losses are greater than generally supposed. The successful introduction of the bag-house, for the

complete collection of the lead fume, marked a metallurgical achievement. The saving of gold, silver, lead, and copper from the slag, by the use of slag-settlers, was a revolutionary step, relatively as important as the bag-house. These settlers will extract 70 per cent. of the silver previously lost in the slag. The bag-house will save losses in smoke to a thousandth of 1 per cent.

Accurate conclusions regarding lead-smelting with blast-furnaces can only be based on yearly records. The longer the period covered the surer will be the deductions. In late years great revolutions have taken place with reference to the blast and other factors, resulting in cheapening the cost of smelting and the saving of values. An important step is comprehended in the lessening of the iron in the slag, necessitated by the scarcity and cost of iron flux. The changed conditions dictated a lessening of the ratio of silica to the combined iron and manganese percentages. Colorado coke, although the percentage of ash in it is high, is particularly adapted to lead-smelting, owing to its peculiar cell structure. *In six years' operations* the average assay of slag was 0.84 ounces silver a ton and 0.53 per cent. metallic lead;

average assay of bullion, 266 ounces silver, 3.49 ounces gold. The composition of the matte showed 77 ounces of silver to each ton, and 0.08 ounces gold; average percentage lead, 10.96; copper, 14.8. The average composition of the slags for the period named, in percentages was: SiO_2, 31.37; FeO, 28.57; MnO, 5.86; CaO, 14.22; BaO, 4.19; MgO, 2.87; ZnO, 6.17; Al_2O_3, 6.10; S, 1.52; CuO, 0.05; PbO, 0.57; total, 101.49 per cent.

The ratio of iron, plus manganese, to the silica was 0.852. The composition of the slag may be more forcefully presented thus: Silica, 31.37; iron bases, 34.43; lime equivalent, 19.75; zinc and alumina, 12.27; small constituents, 2.14; total, 99.96.

The specific gravity of the slag was 3.36 and the matte 4.64. The percentage of matte was 9.03; lead, 12.58; fuel, 13.85.

The daily average tons of ore smelted to the furnace was 125; highest yearly average, 144.

The blast pressure varied from 39 to 47 ounces to the square inch.

The average of flue-dust was 2.43 per cent.; lowest yearly average 1.32 per cent., highest 3.5 per cent.

The quantity of iron flux was gradually lessened, owing to cost. A more silicious and better slag was then produced. An ideal slag, based on experience an observation during the operations noted, would be: 32 per cent. silica; 32 per cent. iron bases; 22 per cent. alkaline bases; 10 per cent. zinc and alumina; 4 per cent. other substances.

The average lime equivalent was 19.75 per cent.; lowest yearly average, 17.76. This was directly occasioned by the zinc and high alumina. The highest lime was (1 year's average) 21.78 per cent. General average of matte, zinc, 6.42 per cent.; barium, 0.73 per cent.; sulphur, 20.29 per cent. iron, 35.00 per cent.; manganese, 1.25 per cent. Slag mixed with the matte caused the apparent presence of manganese in the latter.

The average ounces of silver to the ton of speiss was 24.72; gold, .476, or practically ½ ounce to the ton. The average sulphur in the slags was 1.52 per cent. There was .05 per cent. protoxide of copper in the slag. The average temperature of the slag was 1888 degrees Fahrenheit, equal to 1031 degrees Centigrade. The average tons slag produced in smelt-

ing one ton of ore was 0.95. Gold in slag varied from a trace to 10 cents a ton.

A good idea of the composition of bullion may be obtained from this analysis: Pb, 95.35 per cent.; Sb, 3.27 per cent.; As, 0.28 per cent.; Cu, 0.71 per cent.; total, 99.61 per cent.

ANTIMONIAL LEAD.

A SMALL quantity of antimony is usually contained in the silver and lead ores of the western part of the United States. The Utah and Idaho galenites, containing 50 per cent. to 70 per cent. lead, have antimony in appreciable quantities. When received little or no attention is paid to antimony. They are placed directly on the ore-smelting beds when the silver contents are high. When the sulphur is high and the silver low, they are roasted. The lead produces a compact product, by the formation of a silicate and oxide which binds the particles. Substantially all antimony is concentrated in the bullion by smelting. Some of it goes into the slag, matte, and fume. Its presence in the bullion is rarely manifested, as arsenic and copper also make hard lead. The bullion made at blast-furnaces is weighed, sampled, sent to the refinery, and melted at a low temperature. When the furnace is full there is removed the first copper skimming.

After this has been removed the firing is increased, and there soon arises a light vapory smoke. At a cherry-red heat oily drops appear floating on the surface. These rapidly increase with the increase of heat; soon the entire surface will be covered with antimonate of lead. After firing three hours lessen the heat and open the doors. Cool and remove antimony skimmings. Frequently take a sample with a ladle. Use a mould for the sample. The surfaces are carefully observed as the operation progresses. A white frost-like crystalline coating is seen during solidification. The hardness of the bullion is determined by the size and whiteness of the coating. This coating is characteristic of antimony. As the quantity becomes less, the white, frosty, elliptical spot grows smaller, and *at the moment of complete solidification* the center shrinks, forming a permanent depression. The success of subsequent operations depends on the completeness with which the antimony is removed. There are always two antimony skimmings removed; sometimes three when the bullion is quite hard. The skimmings are essentially antimonate of lead. There is also present considerable oxide of lead,—mechanically

mixed bullion,—some copper, and other foreign substances. When the skimmings have accumulated to 100 tons, they are treated in a small reverberatory furnace with a water-jacketed pan. In this is tamped a coating of brasque composed of ground coke and fire-clay, to give the proper form to the bottom. On this brasque is placed a header course of fire-brick, the bottom sloping to one side, to completely drain the metallic lead. There is the usual fire-box and firing-doors, two charging-doors and two tap-holes, one for the bullion and one for the antimony slag. The slag must be made as free from gold and silver as possible, since any left in it will be lost. In order to free the slag from silver and gold, some reducing agent must be used, such as coal, charcoal, or fine coke. The quantity of metallic lead that drops to the bottom depends on the quantity of fuel used. The fine drops of lead percolating through a layer of antimony slag will wash out gold and silver and they easily alloy with lead. This alloy is tapped into a pot, skimmed, and run into moulds. The bullion is returned to the softening furnaces and treated in every respect like that from the blast-furnaces.

The antimony slag is tapped into pots, cooled, dumped, broken, and stored for six months. Frequently test it for silver and gold. Antimonial lead at one time contained 15 to 25 ounces silver a ton and considerable gold, but this practice has been improved.

Over 600 tons contained: Gold, .08 ounces a ton; silver, 2.45 ounces. To obtain these results mechanical, chemical and metallurgical conditions must be observed.

Prior to the runs have the crucible freed from silver and gold; the fire-brick should be removed if the crucible is old. Time and expense can be saved by removing the 9-inch lining. The furnace walls should be entirely relined with fire-brick, and no portion of the furnace should contain gold or silver.

For antimony runs use slag low in silver. Lead free from silver is used for filling the crucible.

The furnace is blown in on coke and slag, using a low blast. The following antimony charge is recommended: 700 pounds antimony and 500 pounds common slag, 100 pounds limestone, and 200 pounds coke.

It is best not to allow the blast to rise above 10 ounces. The gauge must be exact and sensitive.

The furnace should not be forced for large tonnage, and the top should remain cool. Make on an average 600 bars antimonial lead a shift, each weighing 100 pounds. Use a matte-box holding 125 cubic feet of slag; that passing through it will be remarkably free from silver, varying from a trace to .03 ounces a ton; but the combined antimony and lead in the slag will run 1 to 3 per cent. Re-smelt the pot shells at once. Use a large and deep crucible: inside length at the tuyeres, 140 inches; width, 42 inches; water-jackets, 12 inches bosh. The walls flare outwardly to the feed-door, 17 feet above the centre of the tuyeres. Keep the charge up to the feed-door.

Antimonial lead chills rapidly. The lead-well channel should be 8″×10″. Copper will increase in the lead, and trouble with the lead-well will increase to the end of the run. The channel should be cleaned every few hours by scraping the side walls; put the copper skimming with the slag hoisted for this run.

Generally 30 pounds sulphides on the charge will lessen the trouble from copper by the formation of

matte. After six days the use of sulphides is usually imperative. Gold and silver entering the hard lead constitute a loss. Antimony runs are made once or twice a year. Inhale as little of the fume as possible, as it is poisonous. It may be allowed to escape through a high stack or conducted to the bag-house; the latter course is not advised, since it contaminates the products. The use of a separate bag-house for the fume is advised.

Near the close of the run gradually add copper skimmings from the previous runs and increase the quantity of sulphides when necessary. Finally smelt copper skimmings and all scraps containing antimony. Use finely crushed limestone for blowing out the furnace.

A typical analysis of this product is: Lead, 77.96 per cent.; antimony, 20.99 per cent.; arsenic, 0.62 per cent.; copper, 0.37 per cent.; total, 99.94 per cent.; gold, .08 ounce a ton; silver, 2.45 ounces.

A hard lead containing as high as 49 per cent. antimony may be produced by repeated reduction of lead out of the antimonate of lead. The antimonial bullion is tapped and fresh coal added to the molten mass remaining in the antimony furnace. The

antimony may thus be increased to 60 per cent. Lead with 25 to 30 per cent. antimony is best suited for the market.

ANTIMONY RUNS.

	Average Results in Pounds of Six Runs.					
Charge: Antimony slag	400	700	600	700	700	700
Charge: Common slag	200	450	600	500	500	500
Charge: Limestone	35	120	100	120	100	73
Charge: Coke	100	200	220	220	200	183
No. of charges, 24 hrs.	130	209	200	160	185	204
Bars bullion, 24 hrs.	439	1647	1443	1265	1408	1325
Assay of slag:						
Ag, oz.	0.11	Tr.	Tr.	Tr.	Tr.	Tr.
Pb, per cent.	1.50	1.40	2.0	3.0	1.3	1.7
Analysis of slag:						
SiO_2	34.2	32.6	30.7	31.8	33.0	32.5
Fe	18.4	17.8	21.8	18.1	16.0	15.5
Mn	3.5	2.2	2.4	2.5	2.6	2.8
CaO	21.1	25.0	20.9	22.0	22.7	21.9
Assay of bullion:						
Au, oz.	Tr.	Tr.	Tr.	Tr.	Tr.	Tr.
Ag, oz.	7	5.3	5.5	6.4	4.4	3.5
Hematite (flux)	0	0	80	50	0	0

ROASTING-FURNACES.

Roasting-furnaces have become so necessary an adjunct of all lead-smelting plants that a brief reference to their construction and operation will not be out of place in this work. Its importance, however, may warrant more elaborate treatment in a separate volume.

The most expensive method is by hand roasters. A great many types of patented devices have been invented. Some of them doubtless are an improvement over the so-called fuse-box method, or that style of the furnace where the sulphur is mainly eliminated in the back portion and the charge scraped into the attached fuse-box and fused in one operation.

By reason of the great losses of lead and silver the fuse-box has been practically abandoned. The only claim for its continued use consists in the fact that the fused ore is in better form for treatment in the blast-furnaces, causes an increased tonnage, and

the losses through dusting at the blast-furnaces is practically eliminated.

By a method intermediate between a dry roast and complete fusion, there is obtained what is termed "cintered ore." By cintering the ore and drawing it into pots, the heated mass pounded down by a heavy disk, a product quite compact will result, according to the percentage of lead in the ore, the heat employed, and, chiefly, the degree of compression of the particles.

Good cintered ore does not materially lessen the speed of the furnaces, and by reason of its porosity has many claimants. Indeed, there are not a few metallurgists, having tried one or more of the patented furnaces, who have returned to hand-roasting, because of the great repairs and the loss of time so common to the newer furnaces. The mechanical imperfections will be overcome in time, and hand-roasting will be a thing of the past.

The numerous modifications of the O'Hara furnace are not especially adapted for the lead-smelters. The Wethey-Holthoff is a rational roaster. The Ropp is excellent, and may be brought to a high degree of perfection. The Pierce marks a great step

forward in the art of roasting, and for some kinds of roasting leaves little to bc desired. There has not been as much improvement in the cost of hand-roasting as might be desired. Eventually all the fuse-boxes will be abandoned for cintering furnaces by merely altering the front.

It was accidentally discovered that cintered ore could be improved by punching three holes through the mass in slag-pots immediately after it had been pounded down, and then pushing down through the center a $\frac{3}{4}$-inch steel bar to the bottom of the pot. For some reason unknown, the central portion of the cone will be found quite firm; if this process is omitted, the central portion of the cone will be granular. Much smoke will also escape through these holes, and frequently there will be noticed a condensation of volatile products at the upper edges; particularly is this so of arsenious oxide. This discovery is of considerable importance.

From 1887 to 1899 the chief improvements in roasting consisted in changing from a perfect fusion to a semi-fusion. In fusing, good lump and mine-run coal is necessary. In late years excellent work

has been obtained by entirely dispensing with expensive varieties of coal and using slack.

The cost for hand-roasting varies from $2 to $3 a ton. The cost for roasting on the Brown-O'Hara furnace is about $2 per ton. The labor is cheaper, but owing to its great length, the numerous fireplaces, and the difficulty of regulating the heat the fuel cost is greater than with hand roasters, because more fuel and a higher grade is essential.

The cost of repairs on the mechanical roasters is large, and if this, together with the loss of time, is taken into consideration, it may be found, for a period of five to ten years, that the hand-roasters are actually cheaper.

It is believed that Brückner roasters are destined to play an important part at lead-smelters for roasting the sulphide ores, and that the two great objections to these furnaces, the size of the particles and high percentage of sulphur usually remaining, can be overcome. It is believed, however, the losses by volatilization are greater than with hand roasters.

It seems feasible to use compressed air for consolidating the roasted particles in a heated condition. It is also possible to transfer the roasted ore rapidly

and cheaply to other furnaces, cinter or fuse it, and catch the fume in a bag-house. This belief is based upon experimental data, since it has been shown that the fume arising from the partially roasted ore does not contain free sulphuric acid, and therefore the cloth bags will not be destroyed.

Some mechanical furnaces have reduced the cost of roasting below \$1 per ton, especially upon pure ores high in sulphur.

SMOKE.

THE DRAFT FACTOR.

ON LEAD BLAST-FURNACES.

THIS branch of the subject is mainly a record of actual experiments on the draft. The manometer recorded accurately to the *one thousandths of an ounce pressure*, and was specially designed and constructed for these experiments. Ten feet from the bag-house fan, and at intervals of 10 feet along the blast-furnace flue, eighty-four stations were established, each being numbered. A hole at each station was drilled through the brick wall, and in it was inserted a quarter-inch wrought-iron pipe, to which the manometer was attached. The reading was made two or three times to insure accuracy. Ten feet from the mouth of the fan there was a suction pressure of .35; the same at 180 feet; but back of that the draft gradually diminished from .35 to .24. Pilasters on the back wall caused an obstruction.

By broadening the chamber system the flow of gases was lessened and the deposition of dust increased.

The draft depended on the frequency with which the bags were shaken, on the number of fume-rooms running, and, to some extent, on the weather. When it was moist the draft was not so good. It was influenced by the age of the bags and by the quantity of sulphuric acid in the smoke. When the draft was poor, the furnaces had to be run slow; operators were unnecessarily leaded; feeding of the furnaces was poor; the slag and jackets often became cold, and a train of evils followed. On proper draft and regulation of it is often dependent the success of the entire plant.

The draft in the flue back of the furnaces was not so great. The strongest was .13, and at the extreme end .06 ounce. This is deemed inadequate. A better draft in this particular case would have been of incalculable benefit from a humanitarian, metallurgical, or money-making point of view. *Rarely do blast-furnaces have sufficient draft.* In designing new or altering old works this subject should receive careful consideration. The earning capacity

will be greatly increased by having at all times *a sharp draft on the feed-floor.* Both in the alteration of old and the designing of new plants it will be well *carefully to consider the placing of a fan reasonably near the blast-furnaces, working in connection with the bag-house fans.* When a furnace begins running fast, the inadequacy of the draft at once becomes apparent; cutting the blast to remove the smoke from the feed-floor means a loss of tonnage. By speeding the bag-house fan an improvement in the draft is secured. Overcrowding the fan, however, will minimize results.

The dust-chamber back of the furnaces is generally too small. In a plant of six large furnaces the cross-sectional area should be at least 300 square feet, and the chamber so constructed as to permit of its cleaning without interference with the draft. A simple and cheap way to accomplish this is to have a tunnel under the dust-chamber. The draft of these furnaces was induced by a fan. The draft in the down-take pipes should be at least .10 ounce, and more is desirable. In the dust-chamber immediately back of the blast-furnaces there should be at least .25 ounce. This would permit of fast driving and pro-

duce a clean feed-floor, enhancing profits. Various devices for increasing the draft have been tried without compensating results. The true remedy is in larger fans.

Where fume contains gold, silver, lead, or other elements of value, the stack should not be relied on to produce draft. Fans are recommended.

ON BAG-HOUSES.

In the fan there was a pressure of .32 ounce; in the large pipe opposite one bag-house .26; in the large pipe opposite another bag-house, near the end, .24; in one bag-house .074; in another .076. The loss in the main pipe from friction was .08; loss in the thimble .164; the pressure in the bags was the same as in the fume-room. The loss between the interior of the bag-house and the atmosphere was .076. This shows that two and one half times as much power was required to get the fumes into the bag-house as to filter them.

With the same pressure on the fans more than twice the volume of fume can be filtered in the same filtering area, provided the thimble is large enough to admit the fume as fast as it passes out of the bags.

Item.	Loss in Pressure, Oz.	Horse-power Absorbed.	Annual Cost of Maintenance.
Draft at furnace 2, 4, 5, 7	.031	3.82	$134.52
Down-take from furnace	.022	2.71	97.56
Right angle turn into duct or chamber	.067	8.24	297.64
Duct under feed floor to pipe from No. 9	.020	2.46	88.46
No. 9 furnace and down-take	......	5.70	205.20
Friction through gates	.0083	1.35	48.60
Normal friction through 3 chambers	.0067	1.09	39.24
Friction of pilasters in 3d compartment	.002	.33	11.18
Drawing from 3 compartments to 1	.047	7.66	275.76
Normal friction from entrance of single duct to fan	.0224	3.65	131.40
Friction caused by pilasters	.0171	2.81	101.16
Friction of gate near station 6	.0192	3.13	112.68
Friction caused by curve	.0166	2.60	93.60
Refinery fumes	......	7.40	266.40
Friction to draw to one side at fan	.0355	5.48	197.28
Loss by acceleration	.043	8.04	289.44
Friction in large pipe from fan to thimbles	.080	14.88	535.68
Friction in small thimbles	.164	30.50	1098.00
Friction in filtering	.076	14.14	509.04

These items of expense may be dispensed with: Right angle turn into duct, 297.64; friction of pilasters, 101.16; friction in small thimbles, 1098.00; total, 1496.80.

Experts on moving fluids through ducts lay great stress on having perfectly smooth walls without abrupt turns. The duct through which the fume passed in this instance was not up to the requirements for the most effective work.

Subsequent to the test each compartment of the new bag-house was supplied with large thimbles.

The pressure in all compartments was then again taken, showing, with two exceptions, that the pressure was as great in the fume-room as in the large pipe, being .19 and .20 ounce to the inch.

Prior to the enlargement of the thimbles the fan was exerting 53 per cent. of its energy to draw the fume from the furnaces, and 47 per cent. to dispose of it. The large thimbles reduced it to 62 and 38 per cent., respectively.

ON HAND-ROASTERS.

Hitherto it has been difficult to state with accuracy the proper draft for hand-roasting furnaces. Too sharp a draft has a tendency to materially increase the fuel consumption and quickly remove the heat; it likewise favors the production of a sulphate rather than an oxide. If it be insufficient, the combustion of fuel will not be complete; hence a monetary loss ensues, and the ore will not be properly roasted. By proper regulation of the size of the charge the percentage of sulphur may be varied and a product of 2½ to 3½ per cent. secured. To do this the draft should be near .08 to .09 ounce to the square inch. Where it drops as low as .02 to .03

the fuel is improperly burnt, and the sulphur is much higher. Ores containing much gold or silver should not be roasted too much. The silver and gold in a roasting mixture is, to a certain extent, a guide for the percentage of sulphur that should remain. When one furnace burns more coal than another, similarly situated, the draft should be checked by the use of fire-brick. Sheet-iron dampers are not durable. Interference of the draft is increased by abrupt turns, and by leaving the side and charging doors open unnecessarily long. Other causes include leaks in the sides of the arch; and, frequently, by the improper closing of the doors in the dust-chambers, as well as through the walls.

In one experiment the draft-pressure of the hand-roasters was taken at the fire-box with a Fallis manometer, while the doors were open, and showed .03 ounce to the square inch as an average of six furnaces. None of them had sufficient draft, the flues being too small. The cross-sectional area (inside measurements) was 3 feet wide by 7 feet high. It should be 10 × 12 feet. In the dust-chamber near the furnaces .09 ounce was indicated; near the stack it ran .16 ounce. The flue was known to have

an imperfect draft. The decreased pressure was due to leaks from a defective arch. An accurate manometer will indicate leaks in the flue, obstructions on the inside, and any other irregularities. In another flue the reading was .035 near the furnace, and .16 near the stack. The draft on other hand-roasters, with a different stack, varied from .03 to .10 ounce.

The disparity between a furnace having .10 ounce and another with only .25 ounce draft is quite marked, when is considered the fact that the draft of each is caused by the same stack. A casual inspection disclosed the cause of the trouble, a sudden *rise in the flue*. Hitherto a dip was believed to be bad; this indicated what sharp rises, angles, and turns would produce. The sudden up-rise was then entirely removed and the smoke allowed to flow freely into the stack. This change, simple and inexpensive, produced striking results. The net gain on one furnace was almost threefold. Since there were four furnaces on one side of the stack, and five on the other, some equalization was necessary to have all as nearly even in draft as possible. This was accomplished by inserting a damper.

ON MECHANICAL ROASTERS.

Valuable results were obtained by tests on two mechanical roasting furnaces of the Brown type. The pressures showed a decided inequality of draft. The importance of a damper to control each furnace, instead of one large damper in the common flue, was developed. On the stack side of a common damper was found a draft of .4 of an ounce greater than the hitherto observed in any flue or stack. The draft was nearly twice as strong there as at the stack. In the fire-box of one furnace there was .027 ounce; on another, only .014 ounce, or about half. By the use of the manometer a large obstruction was found in the up-take flues of the imperfect furnace, the removal of which vastly increased the draft. The furnace nearest the stack had the lead. The draft on the upper floor was strong enough, but the heat was too rapidly withdrawn from the down-take flue. By enlarging some openings, throttling others, and whitewashing the outside, the worst furnace was converted into the best. The entry of air to the roasting hearths was regulated and the cause of the trouble removed.

On the furnace side of the common damper there was .17 ounce draft; on the other side, .4 ounce. To correct the fault ⅜-inch steel dampers were inserted on each of the two down-take flues, which removed the smoke from the top floor. By careful regulation of these dampers the fuel bill was materially lessened, the ore was roasted better, and the product increased. Close fitting doors on the sides of each furnace increased the draft. The manometer proved there was much loss of heat through the side walls, and indicated that the *bricks were very porous*. The main flue was 8 feet wide and 5½ feet to the spring of the arch. At a distance in the flue of about 100 feet from the roasters the temperature was 503° F., and a draft of .33 ounce; the velocity was 1420 feet a minute; the volume of gases passing a minute was 56,800 cubic feet. This indicated a tremendous loss of heat. The flue was built under ground, but owing to the extreme velocity little dust was collected. It is important to have a proper draft on all kinds of metallurgical furnaces; it is also quite as essential that the dust-chambers be large. A velocity of 200 to 400 feet a minute is required for the outgoing gases in roasting flues, but this should

never be exceeded. In practice the velocity is generally greater, but where the draft is dependent on a stack, and the flow of the gases is rapid, dust-losses will be correspondingly large. Pits beneath the floor are suggested; but care should be taken not to interfere with the draft. The efficiency of all mechanical roasters is largely dependent on the draft; a fact duly emphasized by these tests.

ON REFINING FURNACES.

It is especially important to have a sharp draft on furnaces used for refining gold and silver. What the draft was on a number of furnaces used for refining bullion by the Parkes' process is here shown. In the fire-boxes of the two softening furnaces .047 and .05 ounce vacuum pressure was found, and .05 ounce in the fire-box of the antimony furnace. In the common flue there was a sharp draft; the average of six determinations taken at various intervals along the flue gave .438 ounce. This is far above the average, and would be too great, unless it were supplied with a regulating damper. In the fire-boxes of lead refining furnaces was found a draft of .034 ounce. This is believed to be ample. There were

eleven Faber du Faur retort furnaces attached to a common flue, and the reading along it varied .13 to .16 ounce to the square inch. The draft increased as its source was approached. The draft of the retort furnaces is quite insufficient, especially where coke is used for heating the retorts, and practical experience teaches that it should be doubled. If it were two or three times sharper the coke would be utilized better; the heat being greater, the operation would proceed much quicker, the recovery of metallic zinc materially increased, and fewer retorts used. These statements are advanced to show the importance of frequent checking of the draft. There should be .25 to .35 ounce pressure back of the cupel and concentration furnaces. While most of the smoke of these was removed, some was lost, and it was found to be rich. *Lengthening the flue will not satisfactorily solve the problem.* The scientific way is to filter the fume through cloth.

The draft in the fire-box of the concentrator was .033 ounce, and of the cupel, .028 ounce. As the heat from these furnaces is great, the flues should be large. The smallest flue should be 4 feet wide, with a head clearance of 7 feet; and from this size

up to 20 feet wide and 25 feet high. *Large flues are not inconsistent with good draft.*

It would be well to have stationed at various places about industrial works a series of delicate manometers. They should be so located as to facilitate frequent readings.

FUME EXPERIMENTS.

ON BLAST-FURNACES.

The loss of solids at the top of blast-furnaces is a very important and far-reaching question. A measured quantity of smoke was aspirated from a hole in the base of the stack, through a disc of muslin held between two large funnels placed top to top. For the tests 76 gallons, or 23.527777 cubic feet, were slowly aspirated, and the total solid contents of the smoke were deposited on the muslin. The weight of the smoke was .0060200875 pounds; hence one cubic foot contained .000255087 pound dust, and as the air entering the furnaces was at the rate of 10,000 cubic feet a minute, the quantity of dust, smoke, and air for seven furnaces was 100,000,000 cubic feet each 24 hours. At the rate calculated a cubic foot, the smoke loss each day of 24 hours was

25,712 pounds, or 12.856 tons. This, at a lead assay of 60 per cent. and a silver valuation of 3.5 ounces, gives 7.7136 tons of lead and 44.99 ounces silver that would be lost in the fume from seven blast-furnaces in one day, or an aggregate of $139,758.50 in one year. When smelting at the rate of 500 tons daily the annual loss of silver and lead will aggregate $150,000 to $175,000 a year. This is not a theoretical calculation, but based on carload lots of burnt bag-house fume, carefully weighed and sampled. *The figures refer to fume, not dust.* The *dust* may be collected in long chambers, but the *fume* cannot. The most satisfactory method for the collection of fume free from sulphuric acid is the bag-house,

The quantity of flue-dust varies 1.5 to 3.5 per cent. of the ores smelted, and this fluctuates according to the size of the individual particles, to the blast-pressure, whether the furnaces are hanging, and possibly to other causes. The size and type of furnace-top influences the quantity of dust produced; the use of charcoal increases it.

The total sulphur in the dust was 6.23 per cent., and a careful examination revealed 0.64 per cent. in

combination as a sulphate; therefore 5.59 per cent. remains as a sulphide.

The most satisfactory results can be obtained by fusing the dust and fume in a reverberatory furnace and collecting the fume at the bag-house. No difficulty is encountered in fusing dust produced by blast-furnaces in any good reverberatory fusing-furnace. The dust collected immediately back of the blast-furnaces always fuses harder than the dust collected some distance from them; indeed it may be stated broadly that both dust and fume fuse easier the further removed from the blast-furnaces. To produce a readily fusible mixture it is best to mix the dust collected near the blast-furnace with that caught some distance from it. The fume collected at bag-houses fuses with great ease, particularly so when burned. A proper furnace for this operation is provided with a deep bottom, a suitable fire-box, and a well regulated furnace-throat. There is no difficulty in fusing 24 to 60 tons a day. The fused product is well adapted to blast-furnaces. Fusion without collecting the resulting fume by cloth is not advised. Dust-chambers of whatever

length will not save this fume, and the fact cannot be too strongly emphasized.

After the bag-house fume has been burnt tnere need be but small mechanical loss. Its fusibility is due to the presence of arsenic and lead oxide.

Experiments covering a period of seven years prove that it is unwise to waste the fume arising from fusing blast-furnace dust. The two most dependable tests are here given: In one case experiment showed an annual loss for one fusing-furnace of $4300; the second gave a net loss on one furnace for one year of $6000. Experiment No. 2 was conducted under conditions more favorable, and the results are therefore believed nearer the actual loss; possibly too low. All prior experiments showed there was an annual loss on each furnace running constantly of $7,000 to $10,000.

A number of experiments were made to ascertain the commercial loss, by actually weighing for a long period the dust entering the furnace, that which came out of it, and taking into consideration the assay value of the raw and fused product. But in every case the result was misleading, owing to the great furnace absorption; also the inability to fix

on the actual dust collected in the stacks, pipes, flues, and chambers; the multiplicity of weights likewise were found fruitful sources of error. The experiment did show, however, that 16 per cent. in weight was lost by fusing. Three experiments proved conclusively that if the burnt bag-house fume is quickly fused in a reverberatory furnace the loss will be quite small. Three other tests showed a daily loss of $9. The average composition of the fume, as a resultant of fusing the bag-house burnt fume, was as follows: Gold, none; silver, 1.75 ounces a ton; lead, 32.4 per cent.; sulphur, 11.0 per cent.; zinc, 3.9 per cent.; arsenic, 0.64 per cent. When the lead percentage diminishes in the bag-house fume the zinc increases, and the arsenic becomes troublesome, because it ignites easily. The burnt fume should occasionally be fused in a reverberatory furnace, and the arsenical fumes allowed to escape, or be caught in a special bag-house. There is no difficulty in saving this arsenical product.

At one time it was thought to treat the burnt bag-house fume by smelting it in the blast-furnace with coke and slag, and allowing it to escape, thus expelling the arsenic. It is true more arsenic is

eliminated in this way than when fused in a reverberatory furnace, yet the lead loss in blast-furnaces is very great if the fume is not saved. When the fume burns well the individual particles are large and firm, the product showing little arsenic, and the percentage of lead 60 per cent. Then it should be put on the ore-beds and smelted along with other ore. To smelt the burnt fume on a blast-furnace, using the maximum quantity on the charge, is not good practice, as shown by two experiments, as follows: Through a hole in the down-take pipe the fume was collected on a muslin filter, using a barrel aspirator. This showed a daily loss of 3.3 tons fume, containing: gold, none; silver, six-tenths ounce a ton; lead, 51.3 per cent.; arsenic, 8.8; sulphur, 6.5; zinc, 7.1.

The financial loss was $63 a day. This expensive mode causes great irregularities, due to rapid growth of wall accretions.

ON ROASTER SMOKE.

Here are given the results of a number of experiments made on roaster smoke.

There was withdrawn from the dust-chamber an

integral portion of the smoke, accomplished with a 6-inch pipe leading to a small fan making 2270 revolutions a minute. This was forced into a cotton bag, and in twenty minutes the sulphuric acid ate holes through it. A woollen bag was destroyed in two and one half hours.

In another experiment quick-lime was selected as a suitable absorbent for sulphuric acid. A box was fitted with wooden shelves, alternately attached to the sides, extending nearly across. A pipe led from the box, and to the end was attached a woollen bag. The quick-lime absorbed part of the sulphuric acid, and the less corrosive gases passed through. The test lasted four hours; the bag was destroyed.

The volume of smoke escaping from two roaster stacks was 75,714 cubic feet a minute, determined by measuring the chamber and ascertaining the velocity with an anemometer. The volume of smoke removed by the fan was 269 cubic feet. Hence there was a distinct ratio between the experimental quantity and the total smoke. These velocities, multiplied by the area in square feet of section, gave the number of cubic feet a minute. The per cent. of total smoke removed by the fan amounted

to .35529 per cent. of $\frac{1}{281}$ part. The average velocity was 1634 feet a minute. Several things combined to make velocity determinations difficult, mainly the high temperature. This is so great that the anemometer should be specially made for the purpose. The average velocity of the gases in the chamber was 529 feet a minute. The velocity in the 6-inch pipe gave, as a mean of three experiments, 1369 feet a minute. The average results of three experiments, calculated in tons lead, ounces silver and gold, and tons dust, with the assay, were as understated: 458.3 tons lead; 10,040.2 ounces silver; 69.55 ounces gold; or, 1140 tons fume lost a year, which gave an average assay of 8.8 ounces silver; .063 ounces gold, and 40.20 per cent. lead. The treatment charges a ton was $8.77. During the experiments the fuse-boxes were running. Later experiments showed the loss was very much greater with fuse-boxes than with cintering furnaces. The average analysis of the dust collected by the three experiments was as understated: silica, 0.87 per cent.; iron, 1.70 per cent.; zinc, 3.87 per cent.; lead, 40.20 per cent.; free sulphuric acid, 5.15 per cent.; combined sulphuric acid, 16.41 per cent.; sulphur

(as sulphides), 0.41 per cent.; arsenic, 4.65 per cent. The experiments proved it is impossible to use cloth to collect smoke from roasting furnaces. They also suggested the use of a solution of sulphide of sodium or calcium, passing the smoke through it to precipitate lead, gold, and silver.

Straw, either coarsely woven or in thick layers, was tried. It was expected the enamel-like coating would withstand the corrosive action of the acid. It was readily acted on, and the freedom from knap showed it to be a poor filterer. The knap on muslin and flannel plays an important part in collecting fume. Several experiments showed that straw simply served as a partial obstructor, retaining not more than 10 per cent. of the solids. Cocoa matting caught 25 per cent. The cost of cocoa, even in the raw state, however, would preclude its use. It was partially destroyed.

The fume was forced through a 2-foot coke column 12 feet high. A tower was constructed of seven 24-inch sewer-pipe tiles. A wooden grating supported the coke, the larger pieces at the bottom, the smaller at the top. An ejector was used capable of discharging 30,000 cubic feet of smoke an hour,

with a pressure of 45 pounds steam. It was started with 80 pounds. The velocity of the smoke through the 6-inch pipe was 1665 feet a minute. The apparatus, therefore, was running at a rate of 22,620 cubic feet an hour. The theoretical estimate of 30,000 feet is made with no obstructions. The coke column was assumed to cause none. After running 24 hours the velocity of the smoke through the pipe was found to be only 444 feet a minute, showing the coke was clogged. On taking the tower apart the bottom was completely obstructed by fume. The coke was moist and adhesive from free sulphuric acid. At a height of 5 feet the coke apparently retained no fume. When the exhaust was working there was a shower of fine condensed steam mixed with sulphate of lead. In the coke was found 2.4 per cent. lead, but since the velocity varied from 1165 to 444 feet a minute, it was impossible to determine the exact velocity. In another experiment seven tiles were used, and the coke was 9 feet high; the velocity of fume 1645 feet a minute. Sulphate of lead still *escaped as a fall of mist in the steam.* In another experiment the velocity showed 1104 feet a minute. This was evidence of the rapid clogging

of the coke. The next day the anemometer showed no current whatever in the pipe. The fact that the fume passed through columns of coke 9 and 12 feet, and only a part caught, showed that little *fume* was collected by the coke; the *material stopped was mainly dust, not fume.* The foregoing experiments indicated the *impracticability of collecting with dry coke fume from furnaces roasting ores.*

A third experiment was made: water was sprayed over the top of the coke. The injector was placed in the inlet pipe. The fume held in suspension in the water passed through the cotton-cloth filterer, demonstrating its impracticability. It would be feasible to use a filter-press, obtaining sulphate of lead in a form easily handled, when it could be charged into blast-furnaces. It might be a vexatious problem how to dispose of the acid water.

The next experiment was running water on the fume, unaided by any solid collecting agent. The apparatus consisted of a tin-box 5 feet high and 24 inches square, provided with projecting shelves alternately from opposite sides, and extending nearly across. These were inclined slightly downward. There were three inlets at the top for water,

and a discharge below. As the smoke entered the bottom 3 jets of water entered the top at the rate of 2 gallons a minute. It trickled over the shelves through the ascending smoke. The smoke was passed through a woollen bag, and the quantity collected was found to be but 10 per cent. of the total fume in the smoke, showing that the action of water collected 90 per cent., and contained solid fume in suspension.

Another experiment made with the same tin box showed a loss of only 5 per cent. The amount of water used in this was 25 per cent. less than the former. The water running from the box had a milky appearance, due to lead sulphate. It appears from the foregoing that *water will abstract solid matter in smoke from roasting-furnaces*. The 5 per cent. loss can be lessened by longer contact with water. Obviously it would seem possible to base an industrial operation for collecting roaster smoke by using water sprays. The corrosive action of the acid, and interference with the draft, are difficulties to be considered. The smoke should not be sprayed with water until it has passed through fans, in order to

prevent corrosion and attachment of fume to the blades.

Many experiments were tried by spraying for filtering the finely divided lead sulphate held in suspension in water. Sand, charcoal, and other porous substances were tried with indifferent success.

Some experiments were made on the collection of fume made by mechanical roasters. It was forced into a flannel bag 28 feet long, at a uniform velocity, for 148 hours. It, as well as the pipe, became hot near the fan, and a *rent several inches long was discovered a few inches from it.* The results, although highly interesting, were allowed to stand for several years; they were then repeated and it was found the fume could be collected for two or three days, sometimes for a longer period; but *under no circumstances would the bag last a week.* It is evident fume of mechanical roasters contains less free sulphuric acid than the hand-roasting furnaces. The sulphates formed by mechanical furnaces are greater than with hand roasters. The numerous fire-boxes of the former emit heated carbon particles and mix with sulphuric acid, reducing

it to sulphurous acid not injurious to cloth. If a sooty flame is maintained at the end of the furnace, there may be little difficulty in filtering roaster smoke through cloth. The same object could be accomplished with fans forcing fume through incandescent carbon. Experiments were made along these lines years ago and failed. Since then, however, gigantic strides in the development of fans and blowers have been made, and it would now seem a simple problem to force roaster smoke through layers of incandescent carbon and collect the fume in a bag-house. The ultimate solution will be in the collection of fume in a *dry state*. Experiments were made with this end in view; on a cintering furnace there was used for settling the fume 170 feet of 20-inch sheet-iron pipe, connected with a 10-inch pipe. The smoke was discharged into a woollen bag 5 feet long and withdrawn from the dust-chamber with a 10-inch Sturtevant fan. The experiment lasted five days; a number of bags were used. A woollen bag lasted on an average four hours, but retarded the flow of smoke. The bags caused the fan to churn the air, and did not deliver properly. *As soon as the meshes of cloth were filled the fabric was*

destroyed. In two experiments 90 pounds of fume were caught, containing: Au, 0.14 ounce; Ag, 13 ounces; Pb, 16 per cent.; SiO, 6.0 per cent.; Fe, 11.0 per cent.; S, 16.0 per cent.; Cu, 0.3; As, 0.7 per cent.; Sb, trace; Zn, 2.5 per cent.; valued at $4700 for one year on one furnace. The cintering furnaces showed an annual loss of $2200 in two other experiments where the fume was filtered through woollen bags. All of this fume was settled in the *pipe, and was not caught by the bags.* Instead of using so long a pipe, in the next experiment only 40 feet of 20-inch pipe was used and the smoke allowed to discharge into the air; hence it was simply a settling and cooling experiment. Sixty pounds of dust were collected. This demonstrated that it would be possible to withdraw the fume from roasters by fans and force it through a series of cooling pipes, radiating the heat, causing a collection of nearly all the fume. A fan or blower could force atmospheric air across the pipes, which, becoming heated, could be used for heating office buildings, drying ore, and other industrial purposes. Great economy would result from the use of this heated air if forced into tuyeres of blast-furnaces. A

temperature could be attained ranging from 350° to 400° F. Air below the freezing-point is often unwisely used during the winter months at many plants. It is possible to handle heated gases, vapors, or fume by fans or blowers. In one instance the temperature was so great the sides of the fan assumed a dull-red color without permanent injury. The blades are not rapidly corroded by heated gases. Numerous experiments at various times have been made with different metals by hanging them in dust-chambers and flues of roasting-furnaces, and there was found scarcely any corrosive action on any of the metals. In some cases the exposure was protracted. It would seem comparatively easy to construct a fan suitable for handling this smoke.

The velocity of the gases at the base of a brick-roaster stack was 850 feet a minute. The stack was 7 feet internal diameter, 80 feet high; the flues were small, and the draft was much impaired by leaks, old age, and poor mortar. In another brick stack 87 feet high and 7 feet diameter, the velocity was 1355 feet. During a number of years the temperature at the base of these stacks ranged from

600° to 750° F. *This accentuates the stupendous quantity of horse-power escaping through stacks.* When the loss of gold, silver, and lead, through highly heated and rapidly moving fume, is thoughtfully considered, the crudeness of many metallurgical operations is emphasized. At the main flue of mechanical roasters a temperature of 1400° F. was found; at the base of the sheet-iron stack the temperature was 1375° F. Where the fume entered the dust-chamber of a hand roaster a temperature of 1650° was observed.

Greater metallurgical losses occur by fusing than by cintering. Fuse-boxes have therefore been discarded. It will be interesting to give some facts verifying this statement. The exhaust-pipes entered the furnace behind the fuse-box. The discharge from the fan led into a bag. There was 136 feet of pipe in the suction, 40 feet in the discharge. The experiment lasted six days, and the bag was not injured. It caught 135 pounds of dust containing: Au, trace; Pb, 45.20 per cent.; As, 3.68 per cent.; SO_3, 15.40 per cent.; Ag, 23.0 ounces; Zn, 6.30 per cent.; Sb, 1.82 per cent.; S, 6.30 per cent. The gross annual loss for each fuse-box was $7780.

The chief metallurgical losses that occur in treating gold, silver, and lead ores are: Loss through smoke and slag; loss through smoke of roasting-furnaces; loss through smoke in refining metals. *The successful installation of the bag-house, for collecting fume from furnaces smelting lead, gold, and silver ores, marks the greatest step taken in metallurgy during the century.*

The losses through the slag are now virtually overcome by the use of slag-settlers. The saving of those passing off in the fume of roasting-furnaces is yet to be achieved. It is a pleasure, however, to supplement the literature on this subject, and to indicate lines of thought that may aid in the solution of the vexatious problem.

Inspection of a number of lead-smelteries in the the Southwest disclosed the fact that little attention was paid to dust-chambers. The erroneous idea prevails that little dust and fume is produced by hand roasters, and that values are small. The Brückner cylinders produce more, but the quantity has been greatly decreased by lessening the revolutions. In operating the Thompson-White inclined cylinder furnace, slightly more than 15 per cent

sometimes passed into the chambers; in the Stetefeldt furnace the loss occasionally reaches 30 per cent. Hand roasters produce less dust than mechanical furnaces. All types of the latter, where the ore is constantly stirred by ploughs and paddles, produce more than Brückner cylinders if the revolutions are not great. Dust-chambers in connection with roasting-furnaces should be at least 12 feet wide and 10 feet high. When constructed of sheet iron, the minimum diameter should be 7 feet; 12 may safely be used.

The smoke should have a velocity of 200 to 400 feet a minute. Slow speed does not lessen the draft of furnaces if the stack or fan be ample. A fan produces the most reliable draft, as heat, cold, wind, rain, and snow does not interfere with regularity. When chambers are large and long, it is possible to save most of the dust, but to save fume from roasters by this method is not possible. Lead losses in roasting argentiferous lead ores vary 2 to 7 per cent.; but they are *largely dependent on the quantity of lead in the charge and the degree of heat employed.* There is an appreciable loss of silver, however, *even under the most favorable conditions. The more intense the*

heat the greater the loss. It is practicable to allow 8 per cent. sulphur to remain in the roasted ore and if the hot material is quickly transferred to a fusing-furnace and melted with a quick, sharp flame, the fume can be collected by straining through cloth.

To roast well, the crushed sulphides should pass through a four-mesh screen. Anything finer is not commended.

Roasting and cintering at a low temperature, with a proper admixture of lead ores, produces a coarse product, but this will not materially check the speed of blast-furnaces.

A table of metals, arranged according to their decreasing affinity for sulphur, follows: Palladium, mercury, silver, copper, bismuth, cadmium, antimony, tin, lead, zinc, nickel, cobalt, iron, arsenic, thalium, and manganese. If the table is used only for metals that concern lead-smelters, the following is obtained: Silver, copper, antimony, lead, zinc, iron, and arsenic.

In the early metallurgy of Colorado partially dried pine wood was used almost entirely for roasting with good results, especially where the arch was high at the fire-box. A brick apron was sometimes

run out from the bridge-wall to prevent cintering the ore by great heat. Later lump coal was used. As competition for ores became sharper and the margins less, a mixture of lump and nut coal, and finally bituminous slack, served every purpose. Ocean steamer grate-bars with $\frac{3}{8}$-inch air spaces are preferable for roasting- and fusing-furnaces. Different grades of coal may be safely used with these. In both hand and mechanical roasting, fuel is one of the chief items of expense and should be carefully watched. The cost of coal for one ton of ore roasted varies from 50 cents to $1.25. Producer-gas can be successfully used for roasting. Ores containing 8 per cent. sulphur have been roasted to 0.2 per cent. with it. Crude oil can also be successfully used for roasting and cintering.

ON REFINERY FUME.

ume arising from furnaces in the Parkes process for separating silver and gold from lead will be here treated. That produced by different methods has the same general characteristics—light in color, often fluffy, and composed of various oxides and sulphates. It can only partially be saved by flues,

Great length necessarily increases savings. The chambers should be wide to lessen velocity; numerous devices have proven inadequate. Filtering this fume through cloth is a complete solution of the problem. Frequently retorts and refining kettles crack, causing appreciable loss. There should be a division of the several varieties of smoke collectible by cloth. There should be a separate bag-house for blast-furnace smoke, and *nothing should be allowed to interfere with it.* Bag-houses should, indeed, be provided for all valuable smoke.

The fume for several of the described tests was collected by aspirating a small portion through cloth held between the rims of two glass funnels. The velocity of the gases was measured by an anemometer, but owing to heat the readings were often difficult and results sometimes problematical. More satisfactory conclusions were reached through the use of a water-motor belted to a small exhaust-fan; it drew the smoke from the flue and forced it through a bag. If temperature will permit, a correct reading and accurate results are obtained. A small electric motor also was used, but found more expensive.

Losses from softening-furnaces were not so large

as from cupel and concentration furnaces. Cupel fumes are heavy, rich, and easily collected. The muslin was uninjured and proved a perfect filterer. The cupel and concentrator fume has been saved at a bag-house. That from softening-furnaces should be allowed to escape through stacks, unless saved by a special bag-house. Tests were made to ascertain the annual fume loss from the combined concentrator and cupel flues. The most reliable showed an annual loss of $3800. Quantity and value of refinery fume varies widely.

ON BOILER SMOKE.

In order accurately to ascertain the chemical composition of solid particles that pass off in smoke of boilers, and at the same time obtain quantitative results, the following experiments were made: The smoke was withdrawn from a flue into which six tubular boilers of 100 horse-power each were discharging smoke. The smoke used was aspirated from the flue by means of barrels of known capacity filled with water and filtered through muslin. By carefully noting the time, the number of barrels of water used, and the velocity of escaping gases, it was

easy to arrive at quantitative results. The six boilers, in burning 30 tons slack coal, lost 0.4 of a ton of soot, which, on ignition, left 5.7 per cent. ash. Of the solids collected, 94.3 per cent. was combustible.

Analysis of the ash of the boiler smoke gave: silica, 67.80; iron, 0.66; sulphur, 0.18; lime, 4.00; magnesia, 0.90; barium, none. As the analysis was made solely for commercial purposes it was not complete. Several tests were made to ascertain if the soot when mixed with flue-dust from lead blast-furnaces could be made to adhere. When 10 per cent. of this soot is mixed with flue-dust the mixture burns slowly, but the combustion is complete; when 15 per cent. or more of soot is used it burns freely. So far as the dust is concerned there was not a sufficient coherence on which to base an industrial operation, but there might be uses to which it could be applied, if saved on a large scale. The feasibility of saving carbon lost in coke ovens has also been demonstrated. The arrangement of the ovens in long rows would render an industrial operation practicable, since the smoke would merely have to be withdrawn by fans, cooled and forced through cloth, as in the manufacture of zinc oxide.

To filter the smoke from large plants in thickly populated cities by the use of cloth, or from several plants in close proximity, should involve no serious engineering problem.

The dingy and uninviting appearance of so many cities, both in Europe and America, is directly traceable to smoke. The idea suggests itself of using fans instead of stacks, filtering the smoke through cloth. Heavy, coarse, unbleached muslin is one of the best filtering substances known, particularly when mixed with wool.

Dr. John Tyndall and other British scientists used both cotton and wool, loosely and in fabric form. Eminent biologists of Europe and America have used cloth for filtering gases to remove particles in suspension. The simplest, cheapest, and most practicable way of purifying air for mines, tunnels, hospitals, theatres, and shops is to filter it through cloth.

Tests with slack coal showed that 1.33 per cent. of solids were lost through the stack. With lump coal the loss would not be so high, and with the hard varieties its loss would be still less.

Basing the coal consumption of Denver Colorado

at 500,000 tons annually and the average loss at .7 per cent., 3500 tons of unconsumed coal would pass into the atmosphere in a year. The coal loss may be ascertained by multiplying the quantity consumed by .7. This closely approximates the percentage of loss through smoke.

Mr. William M. Barr, in a valuable work on combustion, says: "Smoke is regarded as a product of incomplete combustion. In its widest application it is made to include all the products of combustion issuing from the chimney. If the combustion of coal was perfect, the escaping gases would be invisible. Very few analyses of smoke are on record, but from our knowledge of combustion of coal, and of the products of the chemical union of oxygen with its several constituents, we may easily conjecture a qualitative composition of the escaping gases, though the precise quantities of each may be unknown. In smoke there are fine particles of solid carbon which reflect light, and it is this property which renders them visible. If these particles of carbon were not present, the remaining products would be invisible. Whatever that quantity of coloring matter in the smoke as seen escaping from

the chimney, it is to be regarded as so much fuel irrevocably lost. The solid carbon passing off in this manner is often a considerable quantity, *but the actual percentage is almost always overstated.* Sometimes furnaces are so badly constructed that chimneys leading from them are almost constantly pouring into the atmosphere a volume of gases, which, to judge from appearances merely, would seem to be half carbon, or that half the coal fed into the furnace was passing off unconsumed. This mistaken notion as to the percentage of carbon present has been so generally overstated that devices for *burning smoke* have been offered by the most absurd claims in regard to their efficiency as *smoke consumers*, and the great saving of coal to be effected in the event of their adoption. What is needed is not so much a smoke-consuming as a *smoke-preventing* furnace; one which shall be so designed that any fuel rich in hydrocarbon can be completely burned in the furnace proper, or within the chamber containing the incandescent fuel."

Another recognized authority says: "Notwithstanding the prevailing impression as to the great

losses due to the formation of smoke, the actual waste is comparatively insignificant."

This is in strict accord with the experiments noted, and harmonizes with the assertions of Barr; but the author last quoted confirms his position by some carefully conducted experiments made by Mr. J. C. Hoadley. These showed that the total of solid particles passing off in smoke amount to little more than .33 per cent., and that about one-sixth of 1 per cent. of the black smoke is carbon.

These tests have been verified in France by Scheurer-Kestner and Meunier, both of whom found, under different conditions, that the actual carbon lost as smoke particles ranged from .5 to 1 per cent., based on the quantity of coal consumed.

With a knowledge of these facts the reader may safely conclude that the loss of carbon in solid form will be .7 per cent.

The Spanish-American war demonstrated the tremendous importance of forced draft. This feature of the equipment of American vessels was a potent factor in determining the fate of the enemy's fleet. Boilers of war-ships should not emit the slightest trace of black smoke.

THE BAG-HOUSE.

By the term bag-house is meant a structure provided with woollen or cotton-cloth bags for filtering and collecting various kinds of fume. It is customary to force it from the inside to the outside of the bag, 30 feet long and 18 inches wide when distended. Circular bags are strongest and have the maximum surface exposure. It is immaterial whether the gases are forced into the top or bottom, both methods being employed. Where oxide of zinc is manufactured the main delivery pipe should have branches, and on the bottom of the pipe thimbles, to which are attached bags 18 inches in diameter and of varying lengths. These extend down to the floor, and the oxide removed from time to time.

A painstaking examination of various patents and the literature concerning them show that, under proper conditions, lead is quite volatile; that galena ores are readily volatilized; that sulphate of lead is also volatile; that the carbonate, oxide, phosphate (especially chloride, bromide, and iodide) readily go into fume, the solids of which may be recovered. Filtration through cloth or other textile fabric is the

only method the success of which has been fully indicated by long experience and practical tests.

By the term flue-dust is meant all *solid particles* of appreciable size in the products of combustion. It can be settled in long chambers, but neither lead nor any other fume can be settled entirely by any system of chambers devised. Fume is vaporized metals and metaloids, and the individual particles are so small that most of it is lost from the chimney where flues and chambers alone are used.

The *rationale* of the so-called bag-house is to collect volatile matter by withdrawing smoke with an exhaust fan, and forcing it through a textile fabric. The fume is arrested by the cloth, but the gaseous portion passes through it in an invisible state. The tops of the bags are not sewed but pleated, and tied with an ordinary string or wire which serves to hang the bag perpendicularly.

One of the most important advances in metallurgy during the century is the *collection of fume and dust by straining through cloth.* Various patents have been issued covering the use of cloth for the collection of values from smoke. The importance of familiarity with all of these need scarcely be sug-

gested. It was long recognized that lead losses were sustained, and that the quantity varied from 8 to 16 per cent. in a year's operation. The only way to reach a definite conclusion was by practical demonstration—the building of a bag-house. This involved consideration of a multiplicity of details. The equipment included one 200-horse-power engine, two 10-foot fans, two 30-horse-power and two 80-horse-power boilers, one electric light dynamo, and a large water-tank.

Before the bag-house was built tests showed there was lost in 24 hours from 7 to 12 tons metallic lead; subsequent operations confirmed these tests. Indeed the daily saving in metallic lead was greater than shown by experiments. The works, as originally constructed, had only a short flue connecting with a stack 8 feet in diameter and 125 feet high; later there was built an extension of 200 feet, and a return of 200 feet to the stack; this extension was 12 feet wide and 10 feet high, and in a few months paid cost of construction. Owing to the increasing size of the plant it was found that the 125-foot stack was inadequate, and the dust-chambers too short. For a long time it was debatable whether it were best

to build long dust-chambers or a bag-house. Fortunately the latter was adopted.

The foundation already built for the stack was utilized for two 10-foot fans. The main line shaft was driven from the engine fly-wheel by a leather belt; each fan was run by a belt from it, which had a bolted flange coupling, so that either one or both could be run at the same time. Numerous tests showed that nothing was gained by running both, as the air was simply churned and no better suction produced. At a speed of 45 to 52 revolutions the engine produced the best results.

On starting it was found the main principles had been observed, and while some changes were necessary the bag-house was a distinct success. The shafting of the fan was not sufficiently supported; hence longer and stronger journal-boxes were provided. It was deemed best to make a heavy cast-iron pillow-block, and some adjustment of the fan-pulleys were necessary. The pipe leading from the fans was enlarged from 7 to 8 feet. Sharp turns and deflecting plates in this pipe were removed. The inlet to the fans was cramped for handling 220,000 cubic feet of smoke a minute.

A curved brick floor reasonably close to the fan-blades is strongly advised. From the fans to the bag-house the round delivery-pipes should be straight and larger than shown by calculations. A pipe 8 to 10 feet diameter, with frequent circular clean-out holes in the bottom, is preferred. These should be 7 feet apart, measured from the centre, 2 feet in diameter, and closed by tying on a bag of heavy duck cloth; the sacks should be sufficiently long to reach to one foot of the floor, and the smoke prevented from issuing through them by tying together closely near the thimble. This pipe should be No. 8 sheet-steel, and braced with ⅝-inch vertical and horizontal rods. Heavily paint it on the inside as well as on the outside. Ammoniacal gases effect the chief wear of the interior. It is held by two upright rails, placed opposite on an incline, each resting on a stone block 3 feet square and 12 inches thick, flush with the pavement. The pipe rests loosely in a cradle formed by a semicircular bar of iron, $\frac{1}{2} \times 8$ inches wide, riveted to the upper slightly bent ends. The supports should be 12 feet apart,

To the fume-rooms there should be 3 foot circular inlet pipes controlled by dampers. The damper-

rod has a circular disc placed loosely on it. When it is forced in with a vigorous push the disc fits tight against a ground cast-iron ring. It has no tendency to become loosened, owing to pressure of internal gases. There is an open space of at least 14 inches between the fume-room and pipe-thimbles. The opposite ends are connected by a heavy cloth pipe securely wired. It is occasionally necessary to clean the thimbles. If continuous, the pipe will not allow for expansion, and cleaning will be difficult. The value of close-fitting valves is important to health.

Occasionally the fume in the rooms, the pipes, and fans will be found on fire. When such trouble arises shut down the blast-furnace and clean out the fume-rooms and pipes as fast as possible. Deposit the burning fume on the pavements and allow it there to smoulder. These fires do not blaze and produce a black smoke; it is white; at rare intervals it assumes a yellowish tinge when the fume contains much arsenic. This fire *burns without flame.* It may be likened to the burning of punk. Sometimes, when a furnace is blown in or out, the heat becomes so great as to cause the fume

to ignite. As the dust particles become larger the fire travels more slowly and irregularly, and finally goes out altogether. The dust back of the blast-furnaces, because of the size of the particles, and because there is only a small amount of fume at this point, rarely takes fire. It is the *sulphur that burns; the fires are not due to carbon, as there is only a small quantity either in the dust or in the fume.* The heavy dust particles collect near the furnace; the fume at a considerable distance. The fire starts remote from the furnace, creeps backward slowly, and forward rapidly, and then slowly goes out. The resultant dark gray powder is loosely adherent; toward the bag-house it grows harder. The fume that collects in the pipes, 1000 feet from the blast-furnaces, is almost pure sulphide of lead, and *the color is not due to carbon.* To prevent the spread of fire in dust-chambers, build brick cross-walls from 9×12 inches thick and 3 feet high, 200 feet apart, so the product may be cleaned on each side.

The cause of these fires is often obscure:

(*a*) Too great heat from the blast-furnaces: the igniting-point was 350° F., the presence of much arsenic lowers it.

(*b*) A spark from tne blast-furnaces.

(*c*) Spontaneous ignition.

(*d*) Static electricity.

(*e*) Heat generated by the sudden arrest of particles travelling at a high velocity. At an abrupt rise in the pipe fires often originated.

(*f*) Accumulation or confinement of heat. In the bag-house fires invariably start on the side nearest the delivery-pipe, and usually near the bottom of the clean-out door. Incipient fire often was observed by removing one of the doors.

When a fire starts in one fume-room it generally will spread to the others. If the fume is not deep it burns in a loose form, when 3 feet and upwards, in a hard compact form, along the outer edges it has a slightly grayish color. The burnt product is a mixture of oxide and sulphate of lead. Firmness, compactness, and a dark color due to lead sulphide, characterize the centre.

It is best to allow the fume to accumulate to a height of 4 feet in the rooms before ignition. When the heat is not allowed freely to escape, by too infrequent shaking of the bags, fires are more numer-

ous. There should be a sufficient number of bags so that one good shaking in 24 hours will suffice. With insufficient filtering area it sometimes becomes necessary to shake the bags during the night.

All the fume collected at the bag-house is purposely burnt sooner or later. Let it accumulate thirty to sixty days, then close the dampers, take down the door at each end, and set it on fire. When 600 tons of ore is treated in 24 hours there should be three bag-houses, each with 18 fume-rooms 8 feet wide, 15 feet high, and 60 feet long. Leave 60 feet between the bag-houses. Have a repair-shop in close proximity. With ample filtering area fires will rarely occur and will be more easily controlled. When the inlet-pipes are 3 feet in diameter fires become less frequent. A comparison of pressures, using a 2-foot and a 3-foot delivery-pipe to the fume-rooms, is as understated:

	2-Foot Pipe.	3-Foot Pipe.
Draft at gate, near fan	.35 oz.	.45 oz.
Draft at fan	.38 oz.	.51 oz.
Pressure beyond fan	.32 oz.	.41 oz.
Total pressure	.70 oz.	.92 oz.
Volume of fume passing	105,000 cu. ft.	126,900 cu. ft.
Work accomplished by fan	661,500 ft.-lbs.	1,050,632 ft.-lbs.
Power of engine	120	99
Efficiency	16½ per cent.	32 per cent.
Pressure in delivery pipe	0.195 oz.	0.40 oz.
Average pressure of 18 fume-rooms	0.173 oz.	0.341 oz.

The efficiency may be doubled by increasing the diameter of the inlet-pipes from 2 to 3 feet. The unburnt fume at first runs from 70 to 72 per cent. metallic lead; and the burnt fume contains from 65 to 68 per cent. By frequent resmelting, however, on blast-furnaces it will decrease to 30 per cent. lead, due to zinc and arsenic. When the lead in the burnt fume is below 50 per cent. treat it separately by fusing in a deep-bottomed reverberatory furnace, allowing arsenical fumes to escape into the air, or be caught in a separate bag-house. It is surprising how easily burnt fume fuses. Frequently it melts before a fresh charge can be inserted. The fused product is in a compact form for blast-furnaces. It is thin and powerfully corrosive, the oxide of lead eating the furnace-walls. Holding in the furnace such a large mass of molten lead sulphide necessarily occasions much absorption. When it is run into iron trays coated with marl, it will pass through the smallest crack, adhere to the trays, and the spouts soon become corroded, owing to arsenic and antimony. When the lead in the burnt fume is reduced by the presence of zinc, arsenic, and antimony, permit it to accumulate thirty days, when it should be fused. Otherwise

fires frequently occur. As the percentage of arsenic increases the fume is more easily fused and has a lower igniting-point.

In hot weather fires are more frequent. An important feature is an abundance of filtering area. Shake the bags twice daily.

Introducing water when blowing in, blowing out, or dropping down a furnace, is bad; moisture makes the bags clammy and sulphuric acid is formed, which cuts and renders them worthless. A small water spray, however, is necessary to protect the iron. The dust back of the blast-furnaces should be fused. The resulting fume is easily and completely saved in a bag-house; but that from roasting-furnaces, or any other dust or fume containing much *sulphates*, should not be fused and allowed to go to the bag-house, because iron sulphate, by the application of heat, is decomposed, producing sulphuric acid, which destroys the bags.

A few holes at the top of the delivery-pipe will admit light and air, and allow gases freely to escape when a fire occurs. The side doors in the pipe are prevented from leaking by the use of magnesia asbestos mixture. Those of the bag-house are

cemented with a thick magma of unburnt fume. *There is great danger of asphyxiation at the top of the bag-house while in operation.* These gases are principally carbon monoxide, carbon dioxide, sulphurous oxide, and arseniuretted hydrogen, with traces of many other gases, such as sulphuretted hydrogen, marsh gas, and olefiant gas. None of these, except sulphurous oxide, are injurious to vegetation, and the small quantity of this becomes widely diffused with air.

Occasionally there is an unpleasant odor from the bag-house, the cause of which is not apparent. It has a sweet, nauseating smell, and sometimes produces headache.

For the foundation of the bag-house use hard-burned brick; use gray lime mortar to the sheet-iron floor. Especial pains should be taken to fill the cracks in all the walls up to, and slightly higher than, the iron floor. Above the floor-line the building may follow the common mode for other structures; but to the iron floor use three stretchers and a header course. The floors of the fume-rooms should rest on light, strong, brick arches, spanning the entire width. This is necessary to keep the

floors cool. When a deep bed of fume is burned, the side walls and floor become much heated. Hence there should be a sufficient number of rooms, so that five to seven of them may be unused during the burning of the fume without undue strain on the bags. The rooms should remain unused until the floors are cool. Floors that cool rapidly sometimes are made of hollow tiles. Still it is a great desideratum to have a cheap, simple floor, that will radiate heat rapidly. Sheet-iron floors are not serviceable for fume-rooms, neither would floors of cast-iron be good, inasmuch as the sulphide of lead adhere. The surface of the floor should be level, and made of hard, smooth, paving brick. A flat course answers the purpose.

The floor to which the bags are attached is constructed of No. 8 sheet-steel. The bags are tied with 2-inch muslin strips, long enough to go around four times. An expansion joint in the metal floor is unnecessary.

The bags should be arranged in three blocks of twenty-seven each, so the shakers can reach them conveniently. The principle wear takes place near the thimbles and 3 feet above them. It is unneces-

sary to have all made of flannel, because of the cost. But when there is much heat and a considerable quantity of sulphurous and sulphuric acid it is cheaper to use a bag made entirely of flannel. Wool dyed with titanium chloride effectually withstands mineral acids. It is probable that flannel dyed with chloride of titanium will eventually be used for the solution of the roaster fume problem. Titanium stained wool may prove of great benefit at bag-houses, in connection with pyritic smelting, at copper works, and obviously at other places where corrosive fume is produced. The cloth used in a bag-house should be carefully considered. For filtering the fume from lead blast-furnaces use muslin. Any of the different kinds of muslin that have been used by the manufacturers of oxide of zinc will be found sufficient for filtering the fume of lead smelters.

Above the centre of the thimbles there are hooks to which the bags are attached. The upper ends are tied with No. 14 annealed iron or copper wire. The wired bags are hung to the hooks and tied to the sheet-iron floor thimbles.

Bag-shaking is unclean and unhealthy. After a

few years' experience there was originated a peculiar method of protection. The shakers bound their heads with long 2-inch strips of cotton, leaving only the eyes and part of the mouth exposed.

At each end of the bag-houses an inside and an outside ladder, made of oak throughout, should be provided. In designing the most careful attention should be given to ventilation. The rooms should never be cold or damp. Protect the bags from snow, rain, and wind. A covered wooden walk around the outside will be found useful.

Place the lower row of windows 30 inches above the sheet-iron floor and provide heavy shutters. A substantial platform at the head of the steps at each end is an important adjunct. The double doors should be left open at both ends of the loft when the weather permits. Attendants should not linger on this platform because of the noxious gases. Do not run water-pipes or electric wires on the inside. Provide ample light, and along the sides have electric screw plug with flexible cord, and metallic fire-alarm push-buttons at convenient places. Whitewash all inside woodwork, as well as the inside brickwork. All bolts, rods, washers, and exposed

iron should be protected with one heavy coat of graphite paint.

For supporting the roof place a double row of cast-iron posts through the centre. The roof-trusses may be made by bolting 2 × 12 inch plank together. Stones set in the side wall support the trusses.

The roof should have a quarter pitch and be constructed so as to accommodate one-inch sheeting closely laid. On this lay heavy tarred felting and No. 24 corrugated iron, previously painted on both sides with two coats of graphite paint. Break the joints well and have the iron lap two corrugations, using copper nails and lead-washers for the head of each nail. Aside from the main ventilators along the apex it is necessary to construct two additional ventilators, 20 × 10 feet, on each side, using over-lapped wooden slats. During hot weather ventilate from the doors and windows as much as possible. In sunny weather permit a few of the windows to be open; but when dark, cloudy, and damp, keep everything closed, except when the bags are being shaken.

In constructing the bags make two rows of double lock-stitch seam, turning the edges. The thread

passes through four thicknesses. The cloth rarely gives way. The bags, when in service, should not be hard and tight, but have sufficient pressure to keep them well distended; with one hand it should be easy to squeeze the bag into folds, entirely closing off the ascending fume. Generally speaking this may be stated to be essentially sulphide of lead, some sulphate of lead, variable quantities of oxide of zinc, and a minute trace of sulphate of zinc; there is also present ammonia, iron, and manganese sulphates. Arsenious oxide is always present, the quantity variable; it rarely runs under 1½ per cent., and is considered high when it reaches 5 per cent. After burning one of the compartments a white coating may be observed on the surface of the burnt fume. This is composed essentially of arsenious oxide, and often large and beautiful crystals will be seen diffused throughout the entire upper mass. Beautiful specimens of red and yellow sulphide of arsenic have been noticed. Cadmium has not so far been found in this fume. The shades of blues and greens are due to the presence of copper, a considerable product of the bag-house.

Fume in the bessemerization of copper matte can

be completely saved by filtering through cloth. Dry sulphurous acid will not destroy the bags, although sulphuric acid will. A small quantity of sulphuric acid is formed when there is a mixture of sulphurous oxide and water. Much sulphurous acid gas can be handled at the bag-house. The method of fusing dust and fume is rapid and cheap, and the product is in a coarse, lumpy condition for blast-furnaces. Burnt fume can be used to produce an excellent white paint. It generally contains too much gold and silver to allow it to be treated except for these values. A list of fumes that were saved follows:

1. Fusing blast-furnace dust.
2. Fusing roaster fume (not recommended).
3. Fusing burnt bag-house fume.
4. Fusing dust from chlorination works.
5. Fusing dust from cyanide works.
6. Fume arising from slag of blast-furnaces.
7. Fume arising from matte.
8. Fume from blowing air into molten slag from lead-furnaces (experiment on laboratory scale).
9. Fume from bessemerization of lead matte.

10. Fume arising from two large Iles slag-settling furnaces.

11. Fume from production of matte running 35 to 90 per cent. copper.

12. Fume arising from blister copper.

13. Fume from refinery concentrating furnace.

14. Fume from cupel furnaces.

15. Fume from silver-melting furnaces.

16. Fume from gold-melting furnaces.

17. Fume from retort furnaces.

18. Fume from refining furnaces.

19. Fume from lead-melting furnaces.

20. Fume from antimony furnaces.

21. Fume from softening furnaces.

22. Fume from mechanical roasting-furnaces (with disastrous results, as sulphuric acid rapidly destroyed many bags).

23. Fume from blast-furnace producing antimonial or hard lead (not recommended).

24. Fume from blast-furnace melting burnt fume from bag-house (not recommended).

25. *Fume from blast-furnaces smelting all known varieties of gold, silver, and lead ores.*

From the above list will be seen the wide range of

fume already successfully saved by filtering through cloth, although it is often inadvisable to mix too many kinds. There are many metallurgic fumes now going to waste that can and should be saved.

Along the chamber from the blast-furnaces to the bag-house the *dust* particles almost disappear, the deposited material becomes blacker, and particles much smaller; finally it resembles soot, and the percentage of sulphur and lead increases. The dust back of the furnaces may contain as high as 35 ounces silver a ton, yet as distance is reached the *silver grows gradually less and the lead increases*. Particularly at 2000 feet there is 65 per cent. lead and 3 ounces silver in the product.

The fume quietly burns to harder masses as distance from the furnace increases, due to increased percentage of sulphur. It has been customary to burn the pipe fume at a specially constructed brick-house. It is dumped on a brick floor in a pile 3 feet high and set on fire. When burnt the fume is loaded into cars and consigned to the ore bed, where it and other burnt fume from the bag-house is spread in uniform layers. It is a singular fact that this pipe-fume never burns as solidly as the fume that is

set on fire in the bag-house. *The pipe-fume, when disturbed in the slightest manner, never burns as compactly as it would had it been set on fire where originally deposited.* Chemical analysis has not revealed any difference between these fumes, but if the pipe-fume is spread uniformly over the bottom of one of the empty rooms and fresh fume deposited on it the entire mass will burn to a hard, compact mass. For the reasons mentioned it is preferable to put the pipe-fume into one or more of the empty compartments whenever possible, since by doing so a product, harder, firmer, and in better condition for the blast-furnace is produced.

Pave with brick all vacant spaces about the bag-houses, extending the paving to the railroad track. Use hard red brick, laid row-locked on 3 inches sand. Make the joints as close as practicable, and then grout the surface with fresh gray lime mortar, using the broom to thoroughly fill the cracks. A slight admixture of good Portland cement in the mortar is commended. This pavement will soon pay for itself. The brooms used should not be too stiff or heavy. Sweeping this fume is quite an art.

The floors of the fume-rooms should be 8 to 10

inches higher than the railroad car doors, so the dust when loaded into wheelbarrows may be run down-grade to the cars. The difference in level from the top of the rails to the bottom of the dust-room should be 3½ feet. Reasonably close to the track there should be a strong brick retaining-wall about 18 feet from the sides of the bag-house.

Aside from bricking there are two effective ways for handling various kinds of fine material. One is to fuse it in reverberatory furnaces, and the other is to put it in one of the fume-rooms of the bag-house, which has at least two feet of fume on the bottom scattered as thin as possible. The dust from sampling works, roasting-furnaces, blast-furnaces, and indeed many other places, can thus be bound together. Various silver and gold precipitates—products from chlorination or cyanide works or elsewhere — can thus be put into compact form.

The pyrometer is read every hour, night and day, to prevent burning the bags and setting the fume on fire. The pyrometers are loosely attached to the pipe. They can easily be removed and cleaned twice a week. They may indicate a few degrees

above or below the true temperature, yet for all practical purposes are safe guides. Fires are thus obviated. It is best to use a pyrometer both at the fan and at the bag-house.

Provide a door near the mouth of the fan for the introduction of air to cool the gases; open the door gradually when there is a temperature of 300° F. at the bag-house.

It is easier to catch lead-fume than oxide of zinc in bags, because the former is more compact. Copper dust has been successfully filtered through bags, as also the fume arising from the melting of the precious metals. The European type of dust-chamber cannot be pronounced a success. Hanging sheet-iron plates in dust-chambers has now few advocates. Vapor of mercury and retort fume have also been filtered through bags. Water is not a particularly good collector of fume; rape-seed oil is better. Experiments were made with crude vinegar, and that was the best liquid found.

The large loss of zinc at American works could be prevented by filtration through cloth. Prof. John Tyndall filtered the air entering a theatre in London through loose cotton, and collected not only the

dust particles, but also bacteria and various kinds of germs.

Charles T. Williams, professor of chemistry in Delaware College, as far back as 1870, wrote a monograph on the metallurgical losses of the various salts of lead and silver, from which the following excerpt is taken:

"The process of straining the fumes through muslin or fine canvas, which gives such successful results in the collection of zinc fume or oxide, as exemplified by the well-known bag process, has not received from metallurgists the attention its simplicity and completeness merit. From experiments made on a large scale on galena ores, both nearly pure and containing a large amount of blende, the writer is satisfied that its application to the condensation of lead and other metallic fumes would leave little to be desired. It seems strange that it has not been applied to purposes other than the mere saving of oxide of lead, especially when the necessity of a thorough collection of metallic fume is considered."

The author demonstrated muslin cloth used for filtering lead fume should weigh .4 to .7 ounce a

square foot. In a square inch of the cloth there should be 42 threads, both in warp and woof.

The bag-house is a tax on the total cost of ore smelted of 8 to 10 cents a ton.

Experiments made to ascertain the action of heat on muslin bags showed that at 212° F. the cloth was slightly colored, but its strength in no way weakened. At 257° F. the cloth became slightly browned, the strength not appreciably altered. At 302° F. the cloth became much browned and the texture weak, tearing easily.

These experiments were conducted in a closed-air bath, the temperature not varying more than five degrees, and each lasted one and one half to two hours.

Of the total solids passing off in the form of smoke there was found 37 per cent. dust collected by chambers, and 63 per cent. fume collected by bag-house. The *dust* always contains an appreciable quantity of fume; there is little or *no dust* in the fume.

The aim herein has been to show how the stupendous values heretofore lost in smoke can be saved by filtering it through cloth. A simple statement of

the fact might not to every mind be conclusive; but a detailed relation of the evolution of the thought, of the means employed, and the processes used in demonstrating it, should serve to remove any lingering scepticism.

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